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PROCESSING CAPACITY FOR CRITICAL MATERIALS

Charles River Associates, Inc.

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Office of Technology Assessment
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SUMMARY

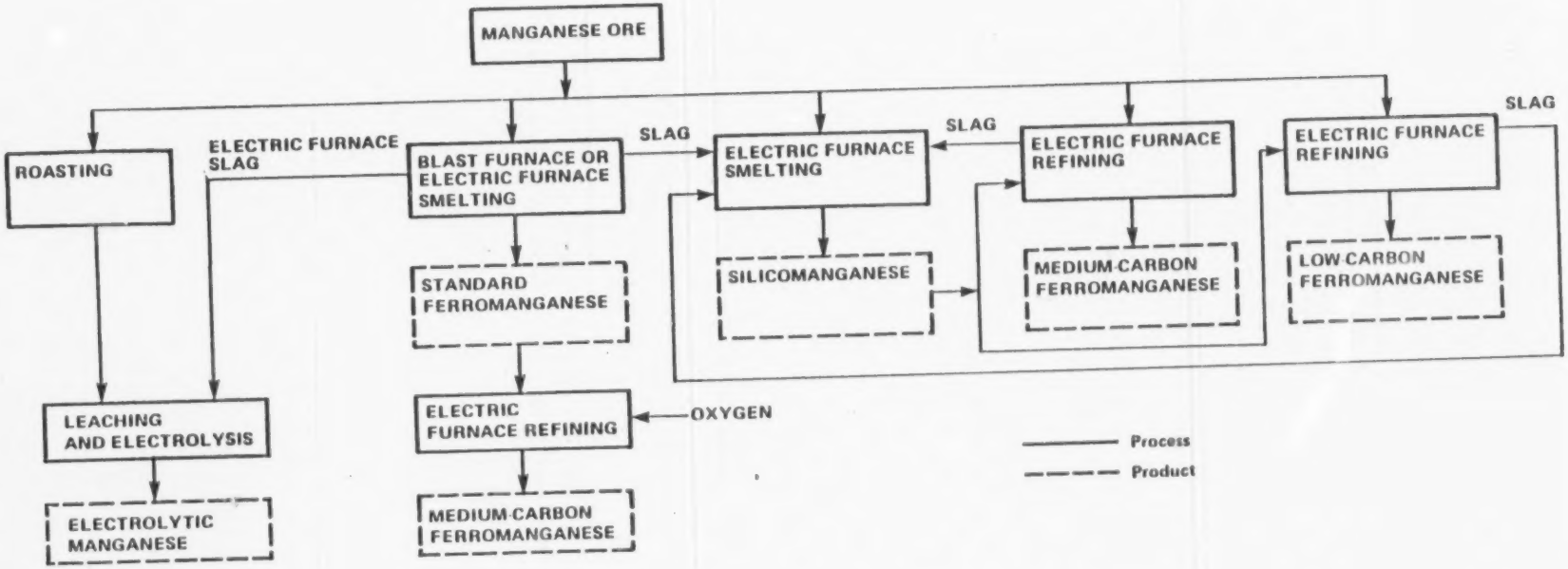
This report examines the distribution and technology of processing facilities for manganese, chromium, and cobalt in the noncommunist world. Manganese is discussed in Chapter 2, chromium in Chapter 3, and cobalt in Chapter 4. The United States depends heavily on imports of all three metals, in either raw or processed form; over the last 30 years domestic mine production of all three has been possible only with government subsidies.

The major processing stages for manganese, chromium, and cobalt, from mine to final processor, are shown in Figures 1-1, 1-2, and 1-3.

The chromium and manganese situations differ in some respects but are essentially similar. Manganese and chromium are primarily used as bulk ferroalloys; that is, as major alloying additives in the production of steels. The United States had sufficient processing capacity for its ferroalloy requirements until the 1970s. Since then the domestic ferroalloy industry has shrunk to the vanishing point. The factors contributing to this decline are essentially the same for both manganese and chromium, and the policies that might be considered to reverse the trend are likewise similar. The cobalt situation, however, is fundamentally different. Cobalt is used primarily in the production of specialized alloys, such as superalloys, tungsten carbides, and permanent magnets. The United States has never had much of a domestic cobalt processing industry, relying instead on imports of refined cobalt (primarily metal, but also oxide and salts). As a result, policies that might encourage the emergence of a major domestic processing industry for cobalt are quite different than for the bulk ferroalloys.

Figure 1–1

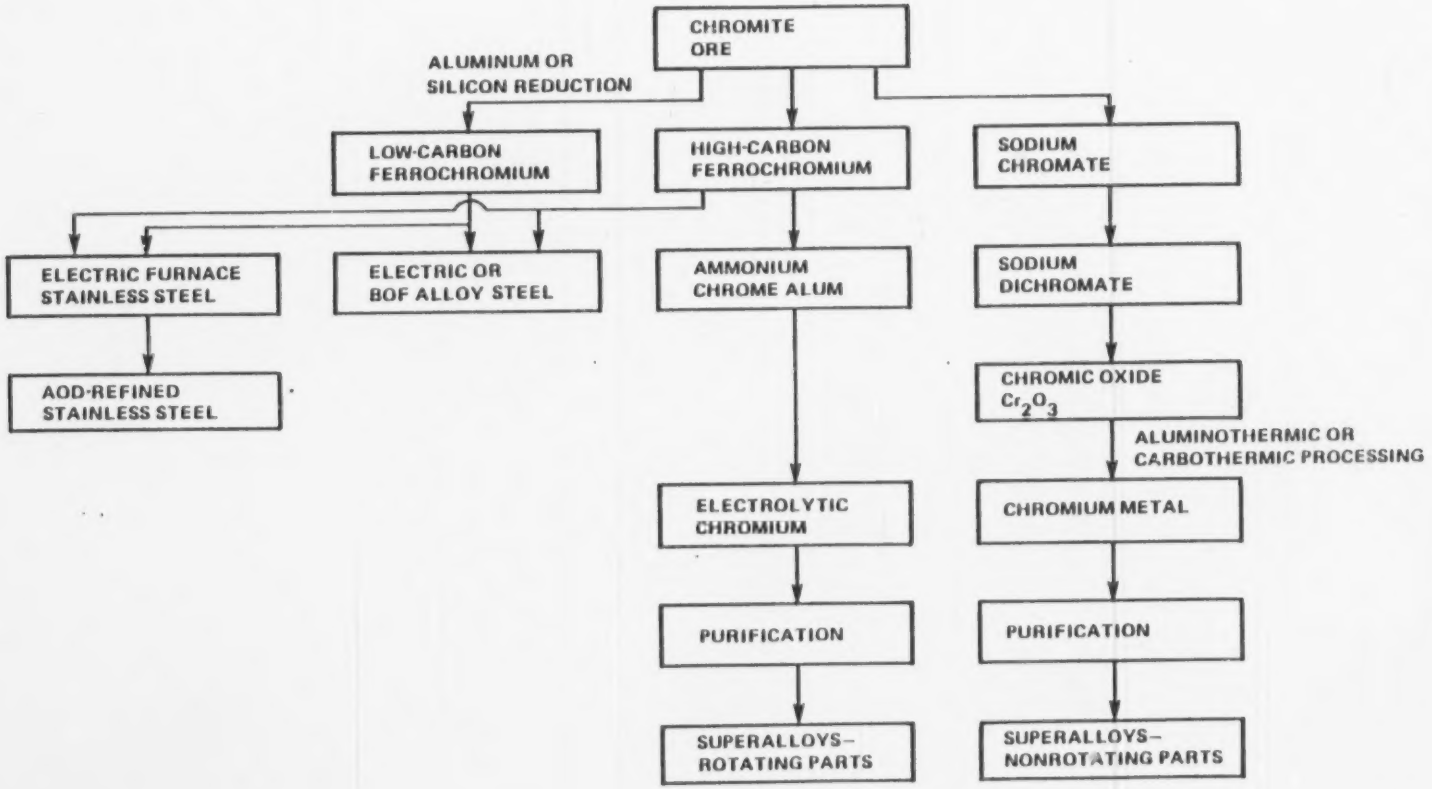
SIMPLIFIED FLOW CHART FOR THE PRODUCTION OF FERROMANGANESE, SILICOMANGANESE, AND ELECTROLYTIC MANGANESE FROM MANGANESE ORE



SOURCE: Charles River Associates, January 1984.

Figure 1–2

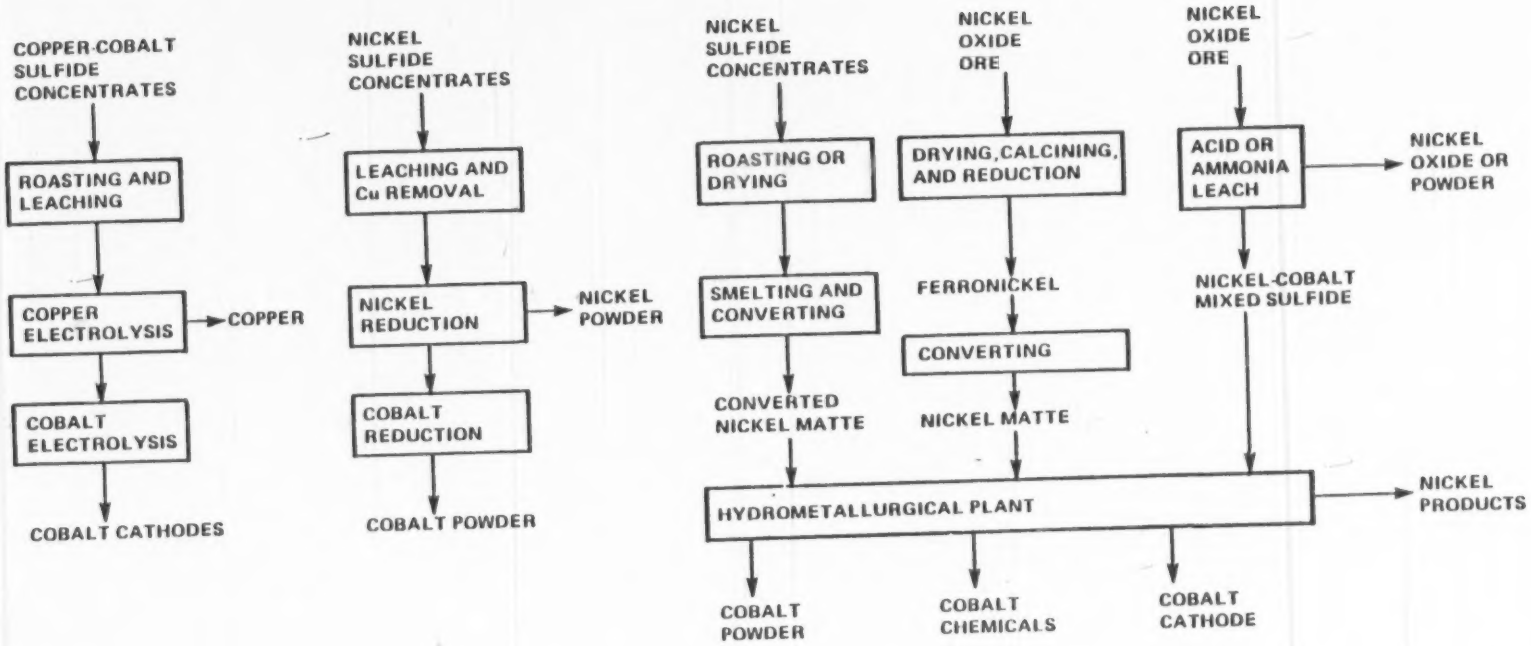
SIMPLIFIED FLOW CHART FOR THE PRODUCTION OF STAINLESS STEELS, ALLOY STEELS, AND SUPERALLOYS FROM CHROMITE ORE



SOURCE: Charles River Associates, January 1984.

Figure 1–3

SIMPLIFIED FLOWCHART FOR THE PRODUCTION OF COBALT



SOURCE: Charles River Associates, January 1984.

MANGANESE

Over 90 percent of noncommunist manganese ore production is concentrated in five countries: South Africa (over 40 percent), Brazil, Australia, India, and Gabon (10 to 15 percent each). Mexico is likely to increase in importance in the future. By far the bulk of the noncommunist world's manganese reserves are in South Africa, but reserves outside South Africa are substantial. The Soviet Union produces manganese on the same scale as South Africa, but East-West trade in manganese is relatively small, with the West currently a net exporter.

Manganese ores generally run 35 to 50 percent manganese, but are concentrated to about 48 percent manganese prior to sale or processing. Most mines are fairly large, except in India. Production in Australia and Gabon comes from single mines. Both open pit and underground mining methods are used in South Africa. The mines in Brazil, Australia, and Gabon are open pit. Because the ores have relatively high manganese contents (compared with base metal ores) and are primarily mined in large, open pit mines, costs per ton of contained manganese are relatively low. Transportation costs are relatively high, compared to the mining costs.

Manganese is added to all steels, to control sulfur levels and to impart specific alloying characteristics. Stainless steels and alloy steels generally have higher manganese contents than carbon steels.

Standard high-carbon ferromanganese (78 percent manganese) is overwhelmingly the most important form in which manganese is consumed. Ferromanganese can be produced in either a small blast furnace, much as pig iron is, or in a submerged-arc electric furnace. Submerged-arc furnaces are used for producing many types of ferroalloys, but they are quite different from the open-arc electric furnaces used for steelmaking. Blast furnace production of ferromanganese is being phased out around the world for cost reasons.

Manganese ore must generally be agglomerated before charging to an electric furnace, by sintering, pelletizing, or briquetting. Lumpy ore does not require agglomeration, but most manganese ores today are not lumpy. Electric furnace production of ferromanganese is a continuous process, with electricity applied continuously. Agglomerated ores and reducing material (coal or coke) can be charged into the furnace either continuously or intermittently. The slags from the process usually have high manganese contents, about 30 to 42 percent.

These slags are often used to produce silicomanganese, another important manganese ferroalloy. Silicomanganese contains 65 to 68 percent manganese and 16 to 18 percent silicon. It is also produced in submerged-arc electric furnaces, using a process similar to that for standard ferromanganese. Furnaces designed for production of standard ferromanganese can be used to produce silicomanganese, and vice versa, but with some losses in production efficiency.

Medium- and low-carbon ferromanganese are much less important than standard ferromanganese on a tonnage basis, but they are needed for many special steels and alloys. Medium-carbon ferromanganese can be produced by oxygen-refining high-carbon ferromanganese, or by reacting silicomanganese with molten manganese ore and lime. Low-carbon ferromanganese is made in a process similar to the latter, but using ferromanganese-silicon (64 percent manganese, 30 percent silicon) instead of silicomanganese. All of these processes use open-arc furnaces, not the submerged arc furnaces of standard ferromanganese.

Manganese metal is used in very small tonnages, but is quite important for specialized alloys. It is usually produced by electrowinning manganese ore dissolved in sulfuric acid.

Two major developments in ferroalloy technology have been considered for ferromanganese. Kawasaki in Japan has investigated a process similar to the basic oxygen furnace (BOF) for steelmaking, in which fine coal, ore, and oxygen are fed through a flash-smelting process. Plasma reduction, using fine ore and either coal or coke as a reductant, has also been considered. Neither technology has been applied on a commercial scale as yet for the production of bulk ferroalloys. Middelburg Steel and Alloys, however, is installing a plasma furnace for charge chrome production in South Africa.

Mining and processing of deepsea manganese nodules is possible around the year 2000, but the economics of manganese recovery from nodules is unclear.

The major factors contributing to the decline of the U.S. ferromanganese industry have been costs. Countries like South Africa have access to cheap power, coal, and labor. Furthermore, exporting ferromanganese rather than manganese ore from South Africa considerably reduces the shipping costs per ton of contained manganese, for two reasons. First, the manganese content of ferromanganese is about 78 percent, rather than the 48 percent in shipping-grade concentrates. Not all shipping and handling costs per ton of contained manganese will be reduced proportionately by the ratio of the manganese contents, but many will be. Second, the 10 to 15 percent of the manganese that is normally lost in processing need not be shipped. These factors combine to give ore-producing countries like South Africa natural cost advantages. These advantages are not by themselves sufficient to encourage processing by the ore-producing countries, however; Australia and Gabon produce essentially no ferromanganese for export. Other factors are also at work, of course. India, for example, has encouraged local production of ferromanganese by at times banning the export of high-grade manganese ores. The South African government has carried out many research programs designed to make use of local materials, through Mintek (formerly the National Institute for Metallurgy). The major factors, however, are costs.

The United States could institute several types of policies to encourage the domestic processing of manganese, but all cost money or have other drawbacks. Trade policies, such as tariffs or minimum import prices, would raise the

cost of raw materials to the domestic steel industry, making it more vulnerable to imports than it is already. Direct subsidy programs are possible, but are difficult to structure efficiently. Tax incentives work only for firms with profits to tax. Having the domestic ferromanganese industry process the manganese ore in the National Defense Stockpile into ferromanganese may well be worthwhile, to support the industry in the short run while guarding against its loss in the long run, but it is essentially a short-run policy; what happens when the upgrade program is completed?

The funding of long-run R&D programs has no drawbacks other than cost, although it will probably not help the ferromanganese industry in its current form. The most attractive projects would be those aimed at recovering manganese in new ways from new types of orebodies, such as from Pacific or Blake Plateau manganese nodules. Thought might also be given to novel mining methods, such as solution mining, or to novel processing methods for low-grade ores.

CHROMIUM

Six countries account for over 80 percent of noncommunist chromium ore (chromite) production : South Africa (over 50 percent), Zimbabwe, Turkey, the Philippines, Brazil, and India. Almost all noncommunist reserves of chromium are found in South Africa and Zimbabwe, but reserves outside those countries are extensive. The Soviet Union and Albania are also major producers. In the past the Soviet Union exported considerable amounts of chromium ore and ferrochromium to the West, but its exports have declined in recent years as Albania's have expanded.

Chromite mines tend to be much smaller than manganese mines, and the grades are lower as well. Typical ores range from 35 to 48 percent Cr_2O_3, which is itself about 68 percent elemental chromium. As with manganese ores, transportation costs are high relative to mining costs.

The lowest-grade ores, under about 40 percent Cr_2O_3, are used primarily in refractory applications, not for metallurgical purposes. The relatively low-grade ores from Finland are an exception, because they also have low alumina contents. Until the introduction in the late 1960s of the argon-oxygen decarburization (AOD) process for making stainless steel, the 40 to 45 percent Cr_2O_3 ores with low chromium-to-iron ratios were also considered unsuited for metallurgical applications. Most of the vast ores in South Africa are of this type and could not be used in metallurgical applications until the AOD process was developed.

Stainless steels are defined as steels with chromium contents above a certain minimum level. On a practical basis, stainless steels contain at least 9 percent chromium, with 14 percent and 18 percent the most common levels. Alloy steels often contain chromium in smaller amounts, ranging up to 1 to

2 percent. Most chromium is consumed by the steel industry in the form of high-carbon ferrochromium, with low-carbon ferrochromium, ferrochromium-silicon, and chromium metal much less important. Before the AOD process was developed, the relative positions of low-carbon ferrochromium and high-carbon ferrochromium were reversed.

High-carbon ferrochromium is produced in a submerged-arc electric furnace in a process similar to that for electric furnace ferromanganese. Lump ore, pellets, or briquets are reduced with coke in the presence of a blend of fluxes. The old standard high-carbon ferrochromium, made from lumpy ores with a high chromium-to-iron ratio, generally had a chromium content of 66 to 70 percent and a carbon content of 4 to 6 percent. Charge chrome, such as that made from the low chromium-to-iron ores in South Africa, generally grades 50 to 55 percent chromium. Low-carbon ferrochromium can be refined from high-carbon ferrochromium or from ferrochromium-silicon.

Chromium metal can be made via an aluminothermic or an electrolytic process. Aluminothermic chromium is suitable for all but the most stringent superalloy applications, for which electrolytic chromium is needed.

Possible new technologies for producing chromium ferroalloys are essentially the same as those for manganese -- flash-smelting in a BOF type process, and plasma smelting. One South African ferrochromium producer, Middelburg Steel and Alloys, is installing the world's first commercial plasma-furnace for ferrochromium.

The factors contributing to the decline of the domestic ferrochromium industry are essentially the same as for the ferromanganese industry -- costs and foreign government policies. Ferrochromium production in South Africa began to increase very rapidly after the AOD process made possible the use of South African ores in stainless steelmaking. Other ore-producing countries, such as Brazil, Turkey, India, and the Philippines, have encouraged the growth of their own ferrochromium industries. The transportation penalties for shipping ore to the United States rather than processing in the country of origin are essentially the same as for manganese.

As a result, the policy problems are similar. Trade, subsidy, and tax policies all have serious drawbacks. Upgrading the National Defense Stockpile, which is already being attempted, is useful but will not maintain the industry after all the ore in the stockpile has been upgraded to ferrochromium. Funding for R&D might benefit the United States in the long run, but it is unlikely to aid the ferrochromium industry in its present form.

COBALT

As mentioned above, the cobalt industry is very different from the manganese and chromium industries. Cobalt is generally recovered as a byproduct of other mineral operations, with over half of the noncommunist world's production coming from copper mining and processing in Zaire. The second largest source is Zambia, also as a byproduct of copper. Other important sources include Finland (also from copper), Canada, Australia, and the Philippines (all from nickel). The only mine operated primarily for its cobalt, in Morocco, shut down at the end of 1982 when its reserves were depleted. The Soviet Union and Cuba are also important producers of cobalt, but the West is a net exporter.

Cobalt operations in Zambia, Zaire, Finland, and Canada are carried out either within companies or through long-standing contractual arrangements. Cobalt intermediates produced in Australia and the Philippines are sent to Japan and, to a lesser degree, Canada, for refining under long-term contracts. Amax in the United States is a small producer, recovering cobalt from nickel matte imported from Australia and Botswana.

Cobalt is generally consumed in the form of electrolytic cathodes, granules, hydrogen-reduced powder, extra-fine powder (for cemented carbide production), cobalt oxide, or other cobalt chemicals. Superalloys generally use high-quality cathodes; they cannot use the hydrogen-reduced powder produced by Amax. Cobalt chemicals are sometimes produced by refiners directly from cobalt intermediates, and sometimes from cobalt metal of almost any kind.

The United States has several plants that process cobalt in some manner. For example, several companies process cobalt metal and oxide into a variety of cobalt chemicals, and several plants reprocess used tungsten carbide cutting materials into usable cobalt products. The two major cobalt processing plants in the United States, however, are the Amax nickel refinery in Louisiana and and the Carolmet cobalt extra-fine powder processing plant in North Carolina. These plants are likely to continue operating into the foreseeable future, although the Amax plant will suffer the usual cycles in the nickel market. No special policies are likely to be needed to keep these plants in production.

The situation is quite different for any other cobalt plants that might be envisioned. Given current supply patterns in cobalt, it seems highly unlikely that the United States can encourage domestic processing of any currently produced cobalt intermediates. It may prove possible for the United States to encourage the recovery of cobalt from new sources for processing in the United States. One example would be the Marcona iron mine in Peru, which contains cobaltiferous pyrites. The United States is funding a feasibility study for the recovery of cobalt from this source. The direct subsidy of domestic production is certainly possible at a somewhat limited scale, but may prove expensive. Funding of R&D could eventually lead to a

cost-effective means of recovering cobalt from domestic low-grade nickel laterites, Missouri lead ores, the Duluth gabbro, and primary cobalt deposits.

2

MANGANESE

INDUSTRY STRUCTURE

ORE PRODUCTION

In 1950 the top five manganese ore producers accounted for 80 percent of Western production: India, 25 percent; South Africa, 22 percent; Ghana, 20 percent; Morocco, 8 percent; and Brazil, 5 percent. In 1982 the top five countries accounted for 93 percent: South Africa, 42 percent; Australia, 14 percent; Gabon, 11 percent; India, 11 percent; and Brazil, 15 percent. The small producers such as Turkey, Greece, Iran, Chile, Argentina, and Thailand collectively have dropped from a share of 11 percent in 1950 to approximately 5 percent in 1982. They have produced at about the same absolute levels through these years, but they have dropped in relative terms.

In this analysis, little is mentioned of the Soviet Union because it seems unlikely that in the event of a disruption in one of the major producing countries, such as South Africa or Gabon, that the Soviet Union would increase production in order to export manganese. In recent years, in fact, the communist world has become a net importer of manganese from the West. Ore production stated in terms of contained manganese is summarized in Table 2-1.

Table 2-1

WORLD PRODUCTION OF MANGANESE ORE: 1950-1978
(Million Metric Tons of Mn Content)

COUNTRY/REGION	1950	1960	1970	1973	1974	1975	1976	1977	1978
South Africa	0.323	0.498	1.220	1.683	1.914	2.274	2.128	1.925	1.662
Australia	0.009	0.028	0.363	0.735	0.730	0.750	1.039	0.710	0.634
Gabon	--	--	0.741	0.988	1.096	1.134	1.057	0.957	1.100
India	0.450	0.536	0.715	0.525	0.521	0.569	0.604	0.576	0.648
Brazil	0.087	0.251	0.506	0.545	0.662	0.621	0.627	0.722	0.608
Mexico	0.014	0.072	0.099	0.145	0.154	0.131	0.129	0.123	0.135
Ghana	0.355	0.262	0.162	0.127	0.103	0.163	0.125	0.137	0.128
Zaire	0.009	0.183	0.167	0.160	0.148	0.148	0.144	NA	NA
United States	0.069	0.047	0.043	0.023	0.032	0.017	0.028	NA	NA
Japan	0.054	0.130	0.089	0.062	0.055	0.052	0.047	0.043	0.036
Morocco	0.141	0.224	0.073	0.102	0.123	0.092	0.082	0.079	NA
Others	0.157	0.339	0.149	0.093	0.125	0.145	0.146	NA	NA
Western World	1.668	2.570	4.327	5.188	5.663	6.096	6.156	NA	NA
U.S.S.R.	1.181	2.055	2.446	2.839	2.848	2.951	2.992	2.957	NA
Eastern Europe	0.041	0.132	0.166	0.146	0.124	0.144	0.143	NA	NA
China	--	0.273	0.300	0.405	0.405	0.405	0.342	0.372	0.504
WORLD TOTAL	2.880	5.030	7.239	8.578	9.040	9.596	9.633	9.2*	9.2*

*Estimate

SOURCE: International Iron and Steel Institute. 1980. "Manganese and the Iron and Steel Industry." Committee on Raw Materials. Brussels. March.

SOUTH AFRICA

The principal deposits are the Cape and the Transvaal. The largest production comes from the Cape deposits in the Kalahari Field, otherwise known as the Kuruman District. The two types of ores mined are the Hotazel/Wessels grade of 47 percent manganese and the self-fluxing Mamatwan grade of 38 percent manganese, named for the mines that produce them. The southern or Postmasburg Field of the Cape Province contains irregular replacement deposits of grades averaging 40 percent manganese. Production is controlled by only two companies, the South African Manganese Amcor Ltd. (SAMANCOR) and Associated Manganese Mines of South Africa Ltd. (AMOSAL). Anglo American Corporation has sold its manganese operations to SAMANCOR. The Transvaal deposits are quite small by comparison, only producing about 200,000 metric tons annually of 42 to 45 percent manganese ore.

Both underground and open-pit methods are used. Extraction recovery of manganese in an open-pit mine such as Mamatwan is approximately 90 percent of in situ manganese. Further handling and beneficiation reduces this to 75 percent. The underground mines, such as Wessels, Gloria, and Middelplatts, operate at a considerable depth and in practice yield only about 65 percent of the in situ ore. That is, of the existing reserves, only 65 percent will ever be brought up to the surface. Further handling and processing losses reduce overall recovery to about 50 percent.

The ore must be transported long distances from the mine head to the ports, 1,200 kilometers to Port Elizabeth and 860 kilometers to Saldanha Bay. The ports themselves operated near full capacity during peak production years. Because of the transportation difficulties and port congestion, any short-term supply response to disruptions elsewhere in the world may be difficult. However, rerouting ore through additional ports such as Durban and Richards Bay could provide relief to the congestion in the short term. The open-pit mines could be expanded up to 50 percent within a year, because additional trucks to haul ore would be the only major requirement for such expansions. The underground mines would take about two years to expand 50 percent because of the time needed to expand the shaft. The underground mines produce at a uniform rate and do not have excess capacity that could be utilized quickly in an emergency. For these mines to expand significantly beyond 50 percent would take on the order of three years. A lead time of at least two to three years would be needed to enlarge the railroad and port capacities to accommodate increased exports from South Africa. The railroad to Saldanha Bay could be doubled on short notice, because it was designed to be doubled. Typical lead times to start a new mine are usually seven years from the initial indication of the manganese ore and about four years if the ore body has already been delineated. The concentration facilities could be expanded at even more rapid rates than the associated mines so it is not the major bottleneck to increased output.

AUSTRALIA

The open-pit mine on Groote Eylandt, situated in the Gulf of Carpentaria of northern Australia, is the only major source of manganese in Australia. Reserve estimates recently were increased from 190 million tons to about 300 million tons of salable product manganese. The 25 to 40 percent ore is beneficiated to a salable product grading 48 percent manganese. Annual production of beneficiated ore is about 2.4 million metric tons. Beneficiation involves crushing, screening, and scrubbing, plus heavy media separation of the lump and optional jigging of the fines. The ore then needs only to be trucked 12 miles to the port of Milner Bay for export. This enhances the logistical importance of Australia in the event of supply disruptions elsewhere in the world.

The Broken Hill Proprietary Company Ltd., runs the mine and has expanded capacity to 3.3 million tons per year of salable product. In the event of a supply disruption elsewhere, Groote Eylandt would be able to double its current supply as fast as new trucks and unloaders could be brought to the island (about one to two years). This would be in line with the time horizon for expanding concentrating facilities. Beyond this, its port would need to be expanded. Groote Eylandt will probably be able to expand significantly in the coming years due to its large reserves, but plans have been shelved currently due to operating levels of only 50 percent of capacity in 1982. Restrictions on manganese ore exports by the Australian government, because of the country's growing steel industry, apparently are not a problem. Labor unrest appears a factor that could cause minor interruptions in supplies from time to time.

GABON

The large open-pit mine at Moanda is the largest manganese mine in the Western world, with annual output of 2.3 million metric tons per year of 51.7 percent manganese ore. Ore reserves are about 190 million tons of product, and another 377 million tons of paramarginal crude ore resources. There are also much larger quantities of low-grade carbonate ores, not considered as part of the resource base at present. The low-grade carbonate ores of Gabon are not likely to be exploited before other marginal oxide reserves. The composition for processing is not as favorable as deposits such as in Mexico, although mining costs should be lower as open-pit extraction for some ore will be feasible. The costs have not been fully assessed but it is known that these deposits are not currently economic. Expansion of annual production to 4 million metric tons of ore is anticipated once the Trans-Gabon railroad is completed by about 1985 (Metal Bulletin, 1980). The transportation network currently is operating well beyond its design capacity, so there would be little response by Gabon to a disruption in the rest of the world.

A further large-scale expansion, such as by 2 million tons beyond the 4-million-ton level, would require expansion of the rail capacity. This might take five to seven years. The original Trans-Gabon railroad will have taken ten or more years to build by the time it is finally completed.

BRAZIL

Brazil currently derives 1.2 million tons per year of salable product from a mine production of 2.5 million tons of ore. Its principal producing mines have problems associated with their location, mine life, or ore quality. The Serro do Navio (Amapa) and the Morro da Mina mines have very limited reserves, 15 million and 3 million tons of product, respectively. The Urucum mine in Mato Grosso State is remotely located, 750 miles inland from Sao Paulo, and its ore has a very high alkali content (3.5 to 4.5 percent). The numerous other mines all are quite small. The Amapa deposit at present rates of production (1 million tpy) is expected to be exhausted in about 9 years. Future increases in production could come from the Urucum underground deposit if manganese prices double.

Resources in Para State include the Azul deposit near the large Carajas iron ore deposit. The transportation infrastructure for Azul will be built only when the Carajas mine is developed, possibly by 1986. The government mining concern, CVRD, was granted exclusive development rights to the Azul deposit in 1982. Azul will produce about 500,000 tons per year of manganese ore, on reserves of 45 million tons of 42 percent manganese ore. Plans are to bring production eventually to the 1,000,000 ton/year level in 1987-1988.

At higher ore prices, five times existing prices, two other mines might be developed: Serra de Buritirama (32 to 47 percent manganese), also near the Carajas field in Para State; and the Ferbasa mine (40 to 42 percent manganese), in Bahia State. The resources are 10 million tons at each site and would produce a combined total of 600,000 to 700,000 metric tons of ore annually within seven years.

INDIA

The reserves total 100 million tons of ore with an average of 40 percent manganese. The quality of the ore is highly variable, from 25 to 48 percent manganese. Actually, the bulk of these reserves are ferruginous manganese ore; 10 million tons are measured high-grade reserves, located in the states of Madhya Pradesh and Maharashtra. These ores contain 30 to 48 percent manganese but are high in phosphorus content (0.14 percent). The present production capacity is about 1.5 million tons of ore with 650,000 metric tons of contained manganese.

The exact capacity is unknown because of the great number of small mines. India's production has become increasingly labor intensive so that capacity

increases at present prices are extremely unlikely and may even begin to decline by the early 1990s. India is having difficulty in servicing its own steel industry so there is the possibility of a severe cutback in exports of ore in the future. The government is currently maintaining an embargo on exports of high grade ore. At much higher prices India probably would reorganize its manganese industry and expand production from its larger reserve base by about 2 million tons of ore annually. A lead time estimated to be five years would be needed for such an expansion.

MEXICO

The Autlan company, which operates the Molango open-pit mine, formerly was linked with Bethlehem Steel but became controlled by private and state-controlled (28 percent) Mexican interests in 1973. The company keeps a reserve base of 13 to 14 million tons which is maintained by systematic exploration. The deposits are carbonate ore of 27 percent manganese content and require calcination to produce a 38 percent manganese salable product of 365,000 tons annually. Molango was expected to double its output in 1981, but this has been postponed due to poor market conditions and financing problems. A further 25,000 tons of higher grade (40 percent) oxide ore are also produced. Expansion in product output is limited, mainly by beneficiation plant capacity, which could be enlarged with a lead time of two years if additional supplies are needed because of a supply disruption elsewhere. The reserve and resource base are very large at Molango with some 30 million tons of proven and 200 million tons of inferred reserves (27 percent manganese) and over 1 billion tons of paramarginal resources of the same grade. Mexico possesses the resources to become a major producer in the future and the only large non-African producer other than Australia or Brazil. Mexico probably can expand production indefinitely at prices near current levels, if need be, although the nature of the mineralization necessitates continual reserve delineation from the known resource base. Part of the problem in reserve delineation is the pinching and swelling of the various stratiform manganese carbonate units of the Chipoco Formation. At Autlan's Tetzintla mine the ore horizon is 25 feet, but may thin to less than one meter in other sections of the manganese horizon. The rugged heavily vegetated terrain that constitutes the region of the Autlan deposits add to the difficulty of delineating proper reserves. Because of this, Cia Minera Autlan maintains an active and extensive exploration program to develop reserves continually.

Mining of these deposits is accomplished by both underground and open pit methods. The degree of mechanization in the underground facilities, and efficient mining methods keep staff and overhead relatively small. The open pit operation, however, results in fairly high cost production with stripping ratios running up to 10 to 1. The positive factors affecting cost include low aluminum content (which permits lower temperature slagging), multiple integrated ferroalloy facilities, favorable low-corrosion ore composition, and efficient Autlan developed nodulizing technology.

GHANA

The Nsuta deposits of high-grade ore (47 percent manganese) are almost exhausted but 30 million tons of carbonate ores still exist. Increased carbonate extraction and beneficiation will take place to keep output at about 350,000 tons per year of product. A roasting/nodulizing plant with a capacity of 350,000 metric tons per year came onstream in early 1983, but due to rail transport problems will not contribute to shipments until 1984. A very gradual expansion to the year 2000 is forecast.

MOROCCO

The Imini deposit has 2.5 to 3.0 million tons of reserves left. Morocco formerly was a more significant producer. The crude ore is 46 percent MnO2 and has been the world's primary source of high quality chemical or battery grade ore in the past. The deposit is expected to be exhausted within 10 years at present annual production rates of 175,000 metric tons of ore.

ZAIRE

The Kisenge-Kamata-Kapolo deposit in southwest Shaba province has reserves of only 1.25 million tons of metallurgical grade ore (51 percent manganese) and 8 million tons of carbonate ore (35 to 40 percent manganese). At recent rates of production of 350,000 metric tons of product annually, and with a recovery in product of 63 percent by weight, Zaire's production will extend only for another four years. Production has slowed down in recent years due to both internal problems in Zaire and the closing of the Benguela railroad in Angola, the only transport route available for the deposit. Any economic exports cannot begin again until the railroad reopens, which could be indefinitely, so material was restricted to internal use. Zaire seeks to utilize its manganese resources by further processing its manganese carbonate ores to finished products for export and domestic use, which would not necessitate shipment through Angola. Alternatively, it is proposed that the dry cell battery factory at Kisenge be restarted to utilize this manganese source.

UPPER VOLTA

The Tambao deposit has some 13 million tons of reserves of 54 percent manganese oxide with an additional 13 million tons of carbonate ore. Extraction awaits the construction of a rail link to the existing railhead, a distance of 350 kilometers. Even though prices probably would have to be twice their current level for the construction of the railroad to prove economically feasible the construction has been undertaken as part of a plan for regional economic development. The railroad would take at least three or four years to build, possibly more. Potential annual output is 600,000

metric tons of ore. The withdrawal of the 4 foreign partners in the project has seriously hindered development, although French aid is still being given for the project.

UNITED STATES AND CANADA

There are no known deposits of manganese ores that can be exploited at current or even substantially higher prices. Table 2-2 lists the potential sources of manganese in the United States and Canada. Of these, at prices even remotely close to current levels, only the Cuyuna Range and Aroostook deposits in the United States and the Woodstock, New Brunswick, and Newfoundland deposits in Canada appear viable. Table 2-3 contains a cost calculation that serves as an example of how we arrived at a required ore price of almost five times the current price to exploit these deposits. The table also shows a similar calculation for the cost of South African ore delivered to eastern U.S. ports. Engineering estimates of mining costs were based on the type of mine, production rate, overburden, stripping ratio, ore grade, depth of an underground mine, relative labor rates in different countries, and other parameters. A 10 percent real return on investment after taxes was assumed in determining the required price of $7.50 (1980 dollars) per metric ton unit (mtu) for typical U.S. manganese ore. Annual output was determined by a 30-year mining life over the extent of the reserve base, with an overall recovery of 50 percent manganese.

FERROALLOY PRODUCTION

The large bulk of manganese ferroalloy production capacity is concentrated in the United States, Japan, and Western Europe, even though there is virtually no ore production in these countries. As Table 2-4 shows, only South Africa as an ore producer is also a very large manganese alloy producer. This total production capacity has been reduced somewhat since the statistics were compiled. Some of the reductions that have occurred in the United States are compiled in Table 2-5 but these generally represent temporary cutbacks.

Since the U.S.S.R. exports virtually no material to the West, its massive production is inconsequential to this analysis. Almost all of the Soviet manganese ore and ferromanganese production is exported to Eastern Bloc countries. The major world manganese producers as summarized in Table 2-1 with the exception of the U.S.S.R. are included in a comprehensive flow chart (Figure 2-1). This flow chart is designed to provide a capsule look at the flow of ore from the major producing countries to market. Quantities are not included in this chart as flows can vary significantly from year to year. To better appreciate the actual distribution of material in the manganese pipeline, a table is presented to determine the origin of ferromanganese and silicomanganese production for the major world producers. Table 2-6 indicates the source of ore for the manganese alloy producers and the net reliance on alloy imports. The distribution of manganese alloy imports by

Table 2-2

MAJOR POTENTIAL MANGANESE RESOURCES WITHIN NORTH AMERICA

Resource	Amount of Ore (Million Long Tons)	Manganese Content		
		Percent	Amount (Million Long Tons)	Iron Content (Percent)
Aroostook County, Maine				
Maple Mountain - Hovey Mountain	256	8.9	23.0	20.7
North and South Districts	80	6 to 12	7.0	17.0
	336	9.0	30.0	20.0
Artillery Peak, Arizona	156	3.9	6.1	3.0
Batesville, Arkansas	175	4.0	7.0	1 to 3
Chamberlain, South Dakota	>2,000	0.6	>12.0	1.0
Cuyuna Range, Minnesota	243	8.0	19.4	32.0
San Juan Mountains, Colorado	43	8.0	3.4	low
Woodstock, New Brunswick	45	11.0	5.0	14.0
Newfoundland	10	10.0	1.0	-

SOURCE: National Materials Advisory Board, Manganese Recovery Technology, Report No. NMAB-323, 1976.

Table 2-3

COST ESTIMATES FOR MINING OF MANGANESE ORE:
A COMPARISON OF THE UNITED STATES WITH SOUTH AFRICA
(1980 Dollars)

	United States	South Africa
Annual Ore Production, Short Tons	1,000,000	1,500,000
Ore Grade (Percent Mn)	10	38
Yield (Percent)	50	75
Type of Mine	Open Pit	Open Pit
Mine Capital Investment (Dollars)	$50,000,000	$45,000,000
Annual Operating Cost	$14,000,000	14,000,000
Unit Cost per Short Ton Recoverable Manganese (Dollars)	280	37.50
Shipping and Loading per Short Ton Recoverable Manganese (Dollars)	200	104
20 Percent Return on Investment before Tax per Short Ton Recoverable Manganese (Dollars)	200	21
TOTAL PRICE (Dollars)	$680/ short ton Mn	$162.50/ short ton Mn
Price per Metric Ton Unit Mn (U.S. East Coast)	7.50	1.56

SOURCE: Charles River Associates, December 1983.

Table 2-4

FERROMANGANESE AND SILICOMANGANESE PRODUCTION CAPACITY, 1979
(Metric Tons Gross Weight)

Japan	1,540
France	700
Norway	645
South Africa	625
United States	615
Germany	335
India	250
Brazil	240
Belgium	230
Spain	170
Italy	154
Portugal	150
Canada	140
Australia	135
Mexico	135
United Kingdom	80
Yugoslavia	45
Argentina	43
Sweden	16
Taiwan	10
Chile	6
Peru	4
TOTAL	6,268

SOURCE: Metal Bulletin Ferro-Alloys Survey. 1979.

Table 2-5

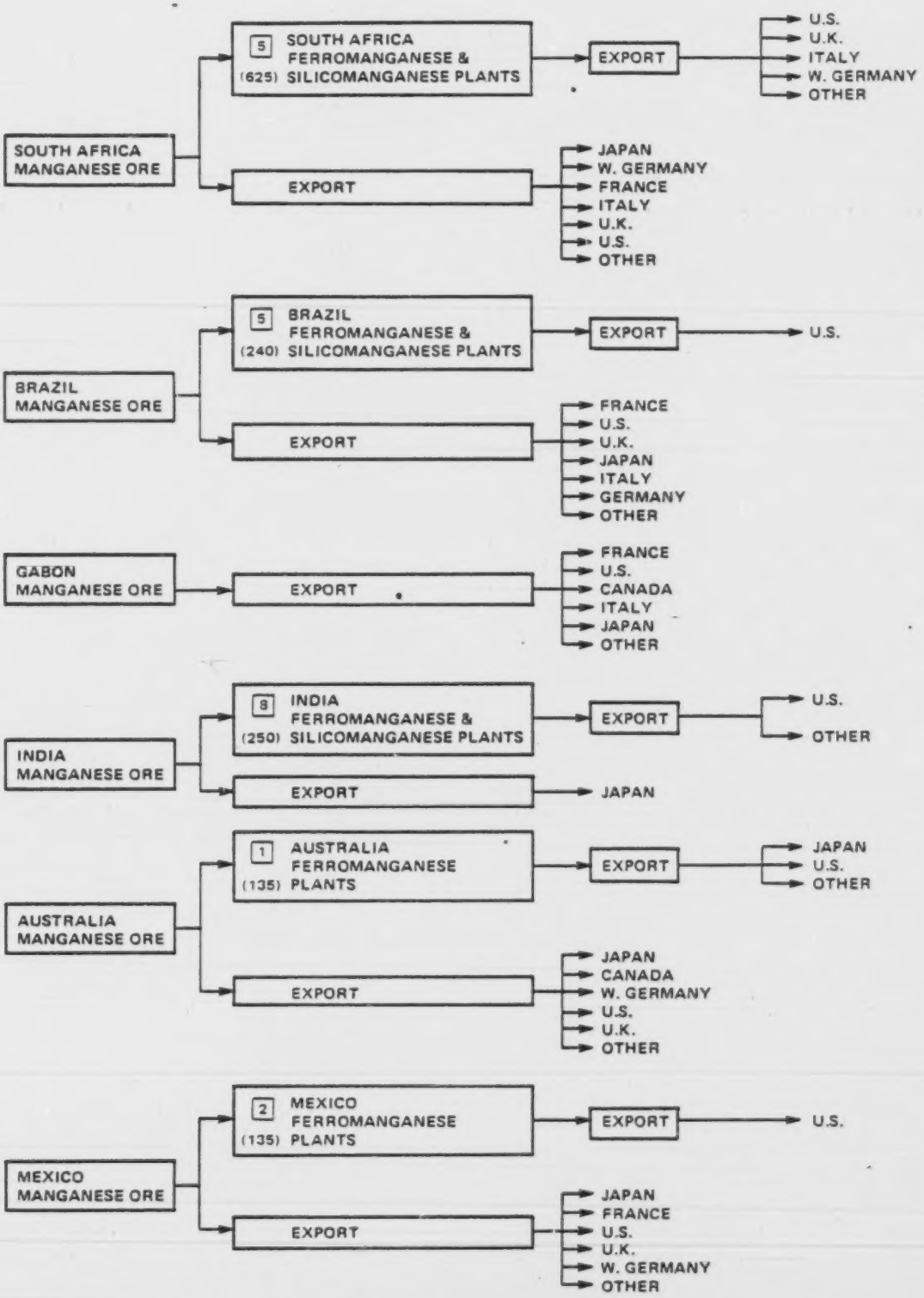
CURRENT STATUS OF U.S. MANGANESE AND FERROMANGANESE PLANTS

Company	Plant Location	Status
Manganese Metal		
Elkem Metals Co.	Marietta, OH	Operating at reduced level
Foote Mineral Co.	New Johnsonville, TN	Shut down since mid-1983
Kerr-McGee Chemical	Hamilton, MS	Operating, but not at capacity
Ferromanganese		
Autlan Manganese Corp.	Mobile, AL	Shut down since mid-1982
Chemetals Corp.	Kingwood, WV	Operating at about 30 percent capacity
Elkem Metals Co.	Marietta, OH Ashtabula, OH Alloy, WV	Operating at low rate
Interlake, Inc. Globe Metal Division	Beverly, OH	Production of ferromanganese silicon
Roane Ltd.	Rockwood, TN	Shut down since 1982
SKW Alloys Inc.	Calvert City, KY	Shut down

SOURCE: U.S. Bureau of Mines and the Ferroalloys Association.

Figure 2–1

FLOW OF MANGANESE ORE, FERROMANGANESE, AND SILICOMANGANESE FROM THE MAJOR MANGANESE ORE PRODUCING COUNTRIES



□ Number of plants ( ) Total Capacity (thousand metric tons of ferro-alloys)

SOURCE: Charles River Associates based on *Metal Bulletin Handbook*. Various issues. *Metal Bulletin Ferro-Alloys Survey*. 1979.

Table 2-6

PERCENT DERIVATION OF FERROMANGANESE-SILICOMANGANESE SUPPLY
FOR MAJOR WESTERN WORLD PRODUCERS, 1978

COUNTRY AND COMBINED SUPPLY OF MANGANESE ALLOYS -- (Thousands of Tons)		SOURCE OF MANGANESE ORE											MANGANESE ALLOY IMPORT SOURCES																
		S. AFRICA	BRAZIL	GABON	INDIA	AUSTRALIA	MEXICO	MOROCCO	GHANA	OTHER	ZAIRE	CONGO	AUSTRALIA	S. AFRICA	BRAZIL	GABON	INDIA	MEXICO	OTHER	US	FRANCE	NORWAY	PORTUGAL	CANADA	YUGA	BEL/LUX	W. GER	SPAIN	CZECH
760	Japan	34	2	3	24	22	3			10																		
690	South Africa	100																										
437	France	38	12	32				3		7																		
390	Norway	46	25	15				4		10																		
437	U.S.	3	6	14		4	2			5			2	26	4		1	3	3		12	5	3	2	3			2
211	West Germany	37	3			6				6				4							11	23				23		5
207*	India				100																							
224	Brazil		97	3						<1																		
97	Belux	16	4						8	5	7	8							7		22	16				7		

Table continued on following page.

Table 2-6 (Continued)

PERCENT DERIVATION OF FERROMANGANESE-SILICOMANGANESE SUPPLY
FOR MAJOR WESTERN WORLD PRODUCERS, 1978

COUNTRY AND COMBINED SUPPLY OF MANGANESE ALLOYS -- (Thousands of Tons)	SOURCE OF MANGANESE ORE											MANGANESE ALLOY IMPORT SOURCES																
	S. AFRICA	BRAZIL	GABON	INDIA	AUSTRALIA	MEXICO	MORACCO	GHANA	OTHER	ZAIRE	CONGO	AUSTRALIA	S. AFRICA	BRAZIL	GABON	INDIA	MEXICO	OTHER	US	FRANCE	NORWAY	PORTUGAL	CANADA	YUGA	BEL/LUX	W. GER	SPAIN	CZECH
73 UK	22	10	1	2				7	5									26		3	21							3
88 Canada	17	19	19						12				10					9	4		10							
100* Mexico							100																					
199 Spain	25	12	15		14			26	8																			
133 Italy	27	7	13		2				4				6					15		20	4				5			

*1977 data.

SOURCE: Charles River Associates based on: International Iron and Steel Institute. 1980. Manganese and the Iron and Steel Industry. Committee on Raw Materials. Metal Bulletin Handbook. 1981.

the major consuming countries indicates that South Africa, Norway, and France are the major Free World sources. The distribution also indicates that only western Europe and the United States are significant importers of manganese alloys. The production numbers of the combined manganese alloys for 1978 are also included in Table 2-6.

The production capacity for Western world manganese alloy capacity as of 1979 is included in Table 2-7. Some of this capacity has been idled since 1979 due to the world economic slowdown but only a small percentage has been permanently decommissioned. The contraction in the world steel industry has led to continued underutilization of manganese ferroalloy capacity. Since new ferromanganese capacity is being added in Brazil, India, and possibly other ore producing countries, this will continue to adversely impact current ferroalloy producers in Europe, Japan, and the United States. Countries with some high cost capacity that may be closed include the United States, United Kingdom, Belgium, and Spain.

CURRENT PROCESSING METHODS

There are two major processing methods for producing ferromanganese alloys: the blast furnace process and the electric furnace process. The blast furnace method is used to produce high-carbon ferromanganese (standard ferromanganese). Until recently, most high-carbon ferromanganese used by the steel industry was produced in a blast furnace process similar to the production of iron; the balance was produced in electric furnaces. Since 1978, the production of high-carbon ferromanganese by the blast furnace method has been discontinued in the United States and Japan and is rapidly declining in Europe (Table 2-8). A discussion of both methods is described below.

BLAST FURNACES

Blast furnaces have been used domestically to produce standard ferromanganese. These furnaces have generally been adapted from old pig-iron furnaces. These furnaces are usually smaller than current pig-iron furnaces and the hot blast temperatures are lower (about 1,000-1,200°F) than for modern iron furnaces. However, the higher hearth-heat requirements for ferromanganese production suggests that the blast temperatures ought to be much higher. Oxygen enrichment of the blast would also probably be economical for ferromanganese, if not for pig iron, production. The main disadvantage of the blast furnace is that the coke requirement per ton of ferromanganese is almost twice that for the electric furnace. Nevertheless, smaller pig iron blast furnaces could readily be converted to produce ferromanganese at minimal capital cost and in a short time. The furnace charge is made up of a blend of manganese ores, coke, and a basic flux such as dolomite or limestone. Additional coke is added to supply the thermal energy for reaction.

Table 2-7

WORLD FERROMANGANESE AND SILICOMANGANESE CAPACITY, 1979
(Metric Tons Gross Weight)

	High-Carbon Ferromanganese	Medium-Carbon Ferromanganese	Low-Carbon Ferromanganese	Silico-Manganese	Manganese Metal	Total All Grades
Argentina						
Electrometalurgica Andina	10,000	--	--	2,000	--	
Carbometal	10,000	--	1,000	--	--	
Grassi	20,000	--	--	--	--	
Australia						
Tasmanian Electro	135,000	--	--	--	--	
Belgium						
Sadacem	150,000	30,000		50,000	--	
Brazil						
Alcan Aluminio	10,000	--	--	15,000	--	
CSN	17,000	--	--	--	--	
Paulista de Ferro-Ligas	--	44,000	--	17,000	--	
Prometal	--	9,000	600	9,600	--	
Sibra	90,000	8,000	--	35,000	--	
Canada						
Chromasco	25,000	--	--	25,000	--	
Union Carbide Canada	65,000	--	--	25,000	--	
Chile						
Carburo y Metalurgia	1,000	--	--	1,000	--	
Manganesos Atacama	4,000	--	--	--	--	

Table continued on following page.

Table 2-7 (Continued)

WORLD FERROMANGANESE AND SILICOMANGANESE CAPACITY, 1979
(Metric Tons Gross Weight)

	High-Carbon Ferromanganese	Medium-Carbon Ferromanganese	Low-Carbon Ferromanganese	Silico-Manganese	Manganese Metal	Total All Grades
France						
Bozel	--	--	--	*	--	
Acieries de Paris & d'Outreau	450,000	--	--	50,000	--	
Pompey	100,000	--	--	--	--	
Sofrem	--	50,000		10,000	--	
Universelle d'Acetylene et d'Electrometallurgie	30,000	--	--	10,000	--	
West Germany						
GFE	--	35,000		--	--	
Peine-Salzgitter	10,000	--	--	--	--	
Hermann C. Starck	--	--	*	--	--	
Thyssen	288,000	--	--	--	--	
India						
Dandeli Ferro-Alloys	12,000	3,000		6,000	--	
Ferro-Alloys Corp.	40,000	--	--	5,000	--	
Jeypore Sugar	24,000	--	--	--	--	
Khandelwal Ferro-Alloys	40,000	--	--	--	--	
Maharashtra Elektrosmelt	50,000	--	--	--	--	
Tata Iron & Steel	30,000	--	--	--	--	
Universal Ferro & Allied Chemicals	36,000	--	--	--	--	
Visvesvaraya Iron & Steel	3,000	--	--	2,000	--	

Table continued on following page.

Table 2-7 (Continued)

WORLD FERROMANGANESE AND SILICOMANGANESE CAPACITY, 1979
(Metric Tons Gross Weight)

	High-Carbon Ferromanganese	Medium-Carbon Ferromanganese	Low-Carbon Ferromanganese	Silico-Manganese	Manganese Metal	Total All Grades
Italy						
Elettrosiderurgica Italiana	60,000	--	--	--	--	
Fucinati	25,000	--	--	--	--	
Italghisa	40,000	--	--	--	--	
Italsider	--	--	5,000	--	--	
Carlo Tassara	5,000	15,000	--	4,000	--	
Japan						
Chuo Denki Kogyo	90,000	--	--	60,000	--	
Furukawa Kogyo	*	*	*	*	--	
Japan Metals & Chemicals	159,100		88,500	162,000	--	
Kobe Steel	50,000	26,000	--	60,000	--	
Minami Kyushu Kagaku	--		15,000	21,000	--	
Mizushima Ferro-Alloy	153,000	--	--	71,400	--	
Nippon Denko	100,000		24,000	95,000	--	
Nippon Kokan	100,000	40,000	--	140,000	--	
Pacific Metals	38,700	12,320	--	22,800	--	
Toyo Menka Kogyo	10,000	--	--	3,000	--	
Toyo Soda Mfg.	--	--	--	--	6,000	
Mexico						
Minera Mutlan	115,000	--	--	--	--	
Ferroalaciones de Mexico	20,000	--	--	--	--	

Table continued on following page.

Table 2-7 (Continued)

WORLD FERROMANGANESE AND SILICOMANGANESE CAPACITY, 1979
(Metric Tons Gross Weight)

	High-Carbon Ferromanganese	Medium-Carbon Ferromanganese	Low-Carbon Ferromanganese	Silico-Manganese	Manganese Metal	Total All Grades
Norway						
Elkem-Spigerverket	120,000	10,000	--	50,000	--	
Sauda Smelteverk	170,000	40,000	5,000	60,000	--	
Tinfos Jernverk	80,000	--	--	110,000	--	
Peru						
Electrometalurgica Nacional	3,600	--	--	--	--	
Philippines						
Ferro-Chemicals	*	*	*	--	--	300 tons (FeMn)
Portugal						
Eurominas	150,000	--	--	--	--	
South Africa						
Delta Manganese	--	--	--	--	28,000	
Electrolytic Metal	--	--	--	--	16,790	
Feralloys	100,000	--	--	--	--	
Samancor	393,000	--	--	52,000	--	
Transalloys	--	10,000	--	70,000	--	
Spain						
Espanola de Carburos	40,000	--	--	20,000	--	
Espanola de Fundiciones Electricos	*	*	*	--	--	20,000 (FeMn)
Ferroaleaciones y Electrometales	--	--	10,000	--	--	
Hidro Nitro	--	35,000		45,000	--	
Sinosa	*	--	--	--	--	

Table continued on following page.

Table 2-7 (Continued)

WORLD FERROMANGANESE AND SILICOMANGANESE CAPACITY, 1979
(Metric Tons Gross Weight)

	High-Carbon Ferromanganese	Medium-Carbon Ferromanganese	Low-Carbon Ferromanganese	Silico-Manganese	Manganese Metal	Total All Grades
Sweden						
Airco Alloys	--	--	--	16,000	--	
Taiwan						
Tung Chou Industrial	*	*	*	3,000	--	7,200 (FeMn)
United Kingdom						
BSC	80,000	--	--	--	--	
Murex	--	--	300	--	--	
United States						
Airco Alloys	100,000	--	--	40,000	--	
Chemetals Corp.	--	25,000	--	--	*	
Foote Minerals Co.	--	--	--	--	11,000	
Kerr-McGee	--	--	--	--	*	
Namco	--	*	*	*	--	
Ohio Ferro-Alloys	50,000	--	--	20,000	--	
Roane Electric	73,000	11,000	--	35,000	--	
Tenn-Tex Alloy		80,000	--	30,000	--	
Union Carbide	*	*	*	*	*	
United States Steel	150,000	--	--	--	--	
Yugoslavia						
Dalmacija	40,000	--	--	5,000	--	

*Capacities not known.

SOURCE: Metal Bulletin Ferro-Alloys Survey. 1979.

Table 2-8

BLAST FURNACE CAPACITY OF HIGH-CARBON FERROMANGANESE*
(Thousands of Metric Tons Gross Weight)

COUNTRY/PLANT	CAPACITY	COMMENTS
FEDERAL REPUBLIC OF GERMANY		
Peine-Salzgitter	50	
Thyssen	250	Situated at Duisburg
FRANCE		
Acleries de Paris & d'Outreau (APO)	350	Not integrated with steel plant The only non-integrated BF producer in the Western world
Pompey	100	
UNITED KINGDOM		
British Steel Corp.	160	Capable of producing 160 tmtpy. Recent capacity utilization has been less than 50 percent
UNITED STATES		
United States Steel		Not in operation at present. Bethlehem Steel used to have a BF facility at Johnstown, but this is no longer operating due to flood damage
South Africa		
ISCOR	50	Part of the Newcastle plant

*Western world only.

SOURCE: International Iron and Steel Institute. 1980. Manganese and the Iron and Steel Industry. Committee on Raw Materials. Brussels. March.

ELECTRIC FURNACES

RAW MATERIAL PREPARATION

Raw material selection is important for satisfactory furnace operation and the efficient production of manganese alloys. The chemical composition of the manganese ore or the blend of ores will, to a large extent, determine the grade of ferromanganese, the slag volume, the slag composition, and the subsequent utilization of the slag. The physical nature of the charge materials in regard to sizing and fines content has a very significant effect on the electric-furnace operating conditions and efficiency. Since manganese ore is the major constituent of the furnace charge, the selection of the ore or blend of ores is of greatest importance.

Blending of ores is desirable and necessary if the required ratios of manganese to iron ore are to be obtained for the production of the specified grades of alloy. In addition to the ore, the coke that is used as the reducing agent has a very significant effect on the results. The coke should be high in fixed-carbon content and low in ash. A low ash content will prevent unnecessary slag formation, which is power consuming.

In addition to the chemical properties of the charge, the physical properties are also very important for a smooth furnace operation. For example, the use of high-grade raw materials with high contents of fines will result in poor furnace operating conditions. Raw materials are required to have a certain sizing and be relatively free from fines so as to permit the free escape of reducing gas through the charge. The size of the ore depends on the size of the furnace and on the time of descent of the charge in the furnace to ensure that the ore is preheated before reaching the smelting zone.

In order to utilize manganese ore fines some form of agglomeration must be used. There are three major types of agglomeration: sintering, pelletizing, and briquetting.

Manganese ore fines can be sintered readily in pans or on a stand type of machine to produce a hard sinter that has good physical properties as a constituent of the smelting charge. In practice a proportion of high quality sinter free from fines results in a lower power requirement for smelting. This is due to the increased porosity of the charge and its beneficial effects on gas distribution.

Pelletizing manganese ore is more difficult and costly than the pelletizing of iron ores. A high firing temperature is necessary, and pellets may tend to fragment during the heating and subsequent cooling.

In briquetting, a small amount of hydrated lime is mixed with the ore, and the briquette is heated at low temperature in an autoclave.

ELECTRIC FURNACE PROCESS

The most common type of electric furnace used for ferromanganese production is the submerged-arc furnace shown in Figure 2-2. The open-arc electric furnace used for producing steel cannot be used to produce standard ferromanganese. The flow of materials in a typical plant is shown in Figure 2-3. The steel furnace shell is normally cylindrical with a flat bottom and is supported on an open foundation that permits air cooling and heat dissipation. The furnace shell's interior walls are lined with refractory or carbon brick at the upper levels and carbon blocks in the hearth.

The electric furnace process is continuous. Electricity is continuously applied to the electrodes, and feed materials that consist mostly of reducing material (coal or coke) and ores may be charged to the furnace on either a continuous or an intermittent basis.

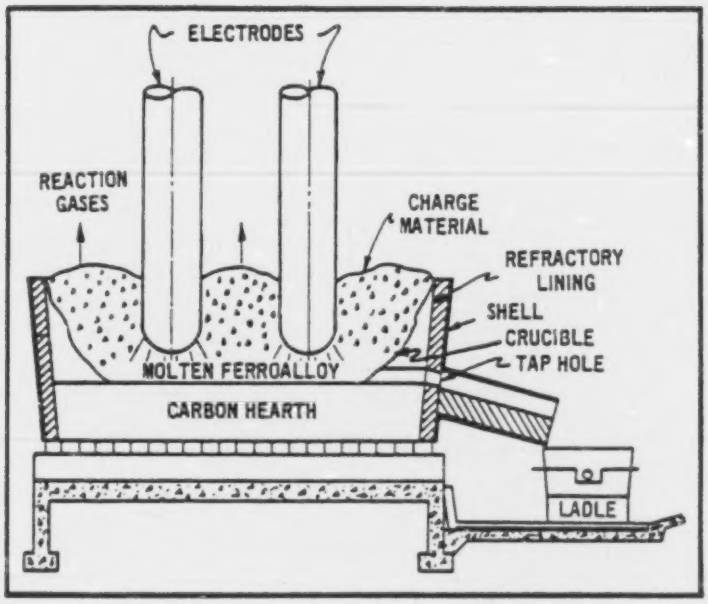
A furnace may not be suitable for more than one product, e.g., a furnace designed for ferrochromium is not suitable for efficient smelting of ferromanganese at the same electrical capacity. The principal difference lies in the electrical conductivity of the slags in the respective smelting processes. Ferromanganese slags contain considerable amounts of MnO and have high thermal conductivities. Thus, the optimum electrode spacing in a ferromanganese furnace is wider than that in a ferrochromium furnace, where the slag is much less conductive, so as to ensure a sufficient resistance. (The lower the resistance, the higher the current has to be; however, high currents and low voltages in AC operation lead to low power factors.) Designing a furnace for ferromanganese production (to achieve the highest possible power factor) means a wide electrode spacing. If such a furnace is used for ferrochromium production, the reaction zones in the vicinity of the electrodes will not overlap and the furnace will not be as efficient as it should be. Nevertheless, a manganese furnace can be operated to produce ferrochromium, even it if it is not the optimum design. One could, for example, adjust the slag composition to decrease the resistance.

Standard ferromanganese (78 percent Mn) production usually results in a relatively high manganese slag which may then be used for silicomanganese production. The manganese content of the slag normally ranges from 30 to 42 percent. Slags are discarded if 1) there is no use for the slag (low manganese), the ore is of low quality, or 2) the ore contains basic oxides (e.g., CaO and MgO) which, if smelted alone, do not permit high manganese slags.

SILICOMANGANESE. Silicomanganese containing 65 to 68 percent manganese and 16 to 18 percent silicon is produced by a process similar to electric furnace smelting of high carbon ferromanganese. The raw materials for silicomanganese production consist of a manganese slag from the production of high-carbon ferromanganese, manganese ore, and/or agglomerated ore, coke,

Figure 2–2

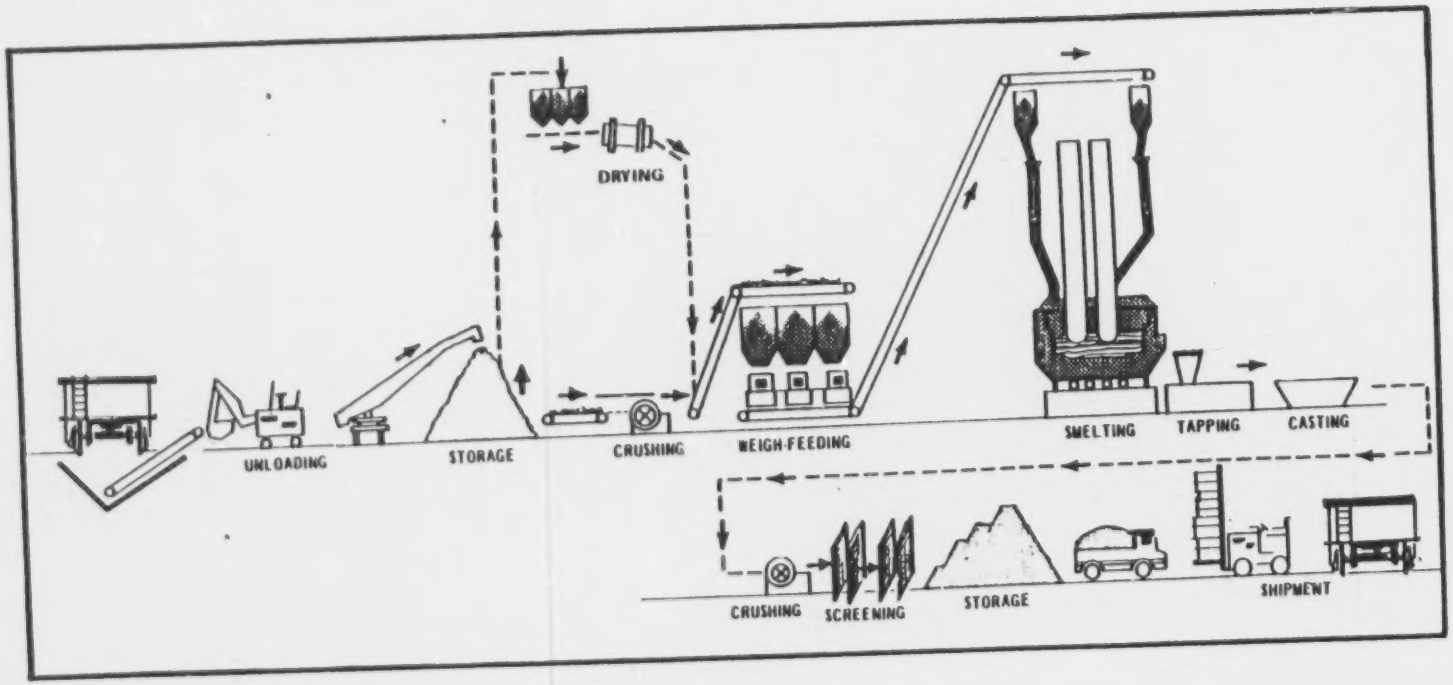
SUBMERGED-ARC FURNACE FOR FERROALLOY PRODUCTION



SOURCE: Dealy, J.O. and Arthur M. Killin. 1974. *Engineering and Cost Study of the Ferroalloy Industry*. Environmental Protection Agency, Office of Air and Waste Management, Office of Air Quality Planning and Standards. Research Triangle Park, North Carolina.

Figure 2–3

FERROALLOY PRODUCTION FLOW DIAGRAM



SOURCE: Dealy, J.O. and Arthur M. Killin. 1974. *Engineering and Cost Study of the Ferroalloy Industry*. Environmental Protection Agency, Office of Air and Waste Management, Office of Air Quality Planning and Standards. Research Triangle Park, North Carolina.

limestone, dolomite, and quartzite. The smelting temperature is higher for silicomanganese production because of the higher resistance of the charge. The electric furnace for silicomanganese is similar to that used for high carbon ferromanganese, but for a given electrical load the crucible, electrode diameter, and electrode pacing requirements are smaller. Provided the gas-cleaning system has the capacity, a furnace designed for high carbon ferromanganese can be operated at substantially higher power when producing silicomanganese because of higher electrical resistance.

LOWER CARBON FERROMANGANESE. Ferromanganese alloys that are not saturated with carbon (medium- and low-carbon ferromanganese) can be produced by refining high-carbon ferromanganese. Medium-carbon (1.0 to 1.5 percent C) ferromanganese can be made by oxygen refining high-carbon ferromanganese or by desiliconizing silicomanganese in a reaction with molten manganese ore and lime. Low carbon ferromanganese is made by using ferromanganese-silicon (64 percent Mn, 30 percent Si) instead of silicomanganese in a similar ore-lime melt reaction. Open arc furnaces are used in such processes.

BLAST FURNACE VERSUS ELECTRIC FURNACE

The electric furnace has many distinct advantages over the blast furnace for the smelting of standard ferromanganese. Electric furnaces are easier to operate, and less coke is required for the process because heat is supplied electrically. In addition, the purity of the alloy is higher, especially when the slag produced is subsequently used in the manufacture of silicomanganese. For low- and medium-carbon manganese and silicomanganese, only an electric furnace can be used, an open-arc furnace for low/medium carbon ferromanganese, and a submerged arc-furnace for silicomanganese.

The selection of either a blast furnace or an electric furnace for the production of standard ferromanganese depends upon local circumstances. For example, an integrated-steelmaker may decide to use the blast furnace process because 1) a blast furnace is available, 2) a supply of low-priced coke can be obtained, 3) supplies of oxygen are adequate, and 4) labor and overhead can be shared with other operations. If these conditions are satisfied, a blast furnace producer can compete with the product from an electric furnace. Under other circumstances, such as for a ferroalloy company, electric furnace production is preferred.

MANGANESE METAL

Metallic manganese (required to some extent in stainless steelmaking and for a number of non-ferrous alloys) may be produced by fused salt electrolysis of prereduced manganese ore (i.e. MnO rather than MnO_2) in an electrolyte containing CaO and CaF_2. More commonly, manganese metal is produced by electrowinning from an aqueous solution of manganese ore in dilute sulphuric

acid. Solution purification is necessary and the cells employed use diaphragms around the anode.

ALTERNATE PROCESSING METHODS

While the submerged-arc electric furnace process predominates in the production of ferroalloys, other methods are being explored. Kawasaki, in Japan, is reported to be piloting a reactor, similar to a steelmaking BOF, to which fine coal and ore plus oxygen are fed through a flash-smelting burner; no electrical heat is necessary but the process depends, economically, on the use of CO produced in rereduction or electrical cogeneration.

A quite different technique is plasma reduction, using fine ore and coal or coke as reductant. A diagram of a plasma furnace is shown in Figure 2-4 for a plant producing charge chrome. In both this case and the Kawasaki process a major advantage is that the ore and coal need not be agglomerated.

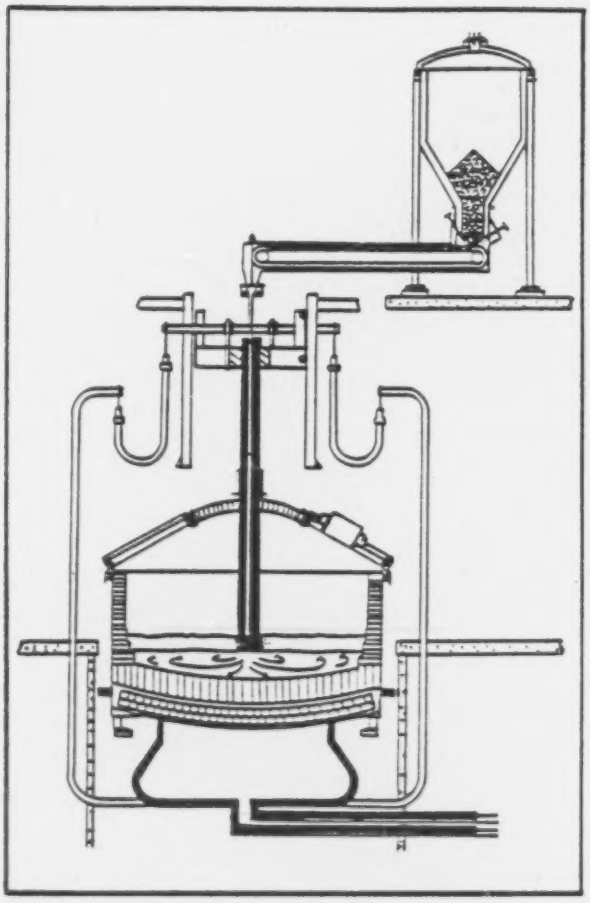
The plasma furnace is basically an electric arc furnace in which the carbon electrodes are replaced by metallic electrodes and the electric arcs by plasma arcs. The basic design would be similar for ferromanganese production. An essential difference in design is the installation of plasma torches in the wall, and not in the roof of the furnace as with carbon electrodes. Both hollow carbon electrodes and nonconsummable water cooled copper electrodes may be used. The major advantages claimed for the plasma furnace process are as follows:

- Increased economy, as electrodes are barely consumed;
- Fewer environmental problems, as less dust and waste gas are generated, and the noise level is extremely low;
- No need for expensive agglomeration processes for fines, and higher product quality (e.g., a lower carbon content);
- Fine coal can be used instead of the more expensive coke; and
- Increased yield due to lower slag production.

In the production of ferroalloys the plasma furnace can either be used for melting metallic fines or for reduction of ore, particularly in the form of fines. Voest-Alpine of Austria has supplied a plant for the first application to a leading ferroalloy producer. It is designed to melt metallic ferromanganese fines and was due to start up in March of 1983. A plasma furnace used for the production of ferromanganese can be used to produce ferrochromium or other ferroalloys.

Figure 2–4

DIAGRAM OF MIDDELBURG STEEL AND ALLOYS' DC ARC FURNACE AT KRUGERSDORP, SOUTH AFRICA



SOURCE: *Metal Bulletin Monthly*, 1983, "Middelburg Strives to Stay Ahead in Charge Chrome," July.

Plasma arc reduction processes have occasioned a good deal of interest, but they have yet to be demonstrated on a production scale for ferroalloy production. They would seem to merit attention because the high-intensity heat source would permit economical operation on smaller-scale units (at least as compared with large, submerged arc furnaces) but it has yet to be shown that they can compete on the basis of kwhr/ton produced. (See Journal of Metals, 1980, Chang and Szekely, 1982, and Rykalin, 1980.)

Mining and processing of deepsea manganese nodules is possible in the long run, but probably not before 1995 or 2000. Manganese nodules are discussed in Appendix A.

DEMAND PROJECTIONS

Manganese demand is closely tied to steel production, so we expect it to grow slowly over the next 25 to 30 years. The demand for steel in the United States will grow much more slowly than it did over the last 30 years, due to the continuing evolution of the U.S. economy from an industrial base to a service base. Competition from imported steel will likewise continue at a high level. Furthermore, the gradual increase in the use of continuous casting will reduce scrap generation and hence production requirements per ton of steel shipped. On the other hand, the dollar should weaken in value somewhat over the next few years, considerably improving the competitive position of the domestic steel industry over its disastrous state of 1982-1983. Carbon steel production in the United States is expected to grow about 0.5 percent annually in the long run, using 1981 as a base year. Stainless steel production is expected to grow much faster, at an average rate of 4 percent annually from the base year of 1981. Production of high-strength low-alloy steel is projected to grow at 3 percent annually, and production of other steels at 2.5 percent annually.

Manganese demand projections are summarized in Table 2-9. Carbon steels are by far the largest end use for manganese and will continue as such for the foreseeable future. Alloy steels will grow in relative importance, particularly because they have high manganese contents, compared to carbon steels. The consumption of manganese ore in iron and steel production (primarily blast furnace additions) will grow at about the same rate as total manganese demand in steels. The demand for manganese ore in chemicals and miscellaneous uses will grow at about the same rate as industrial production, expected to be 3 percent in the long run.

FACTORS AND POLICIES AFFECTING U.S. PROCESSING

Many factors have contributed to the decline of the U.S. ferromanganese industry over the last decade. First, two of the largest ferromanganese producers in the United States ten years ago (U.S. Steel and Bethlehem Steel)

Table 2-9

PROJECTIONS OF U.S. MANGANESE CONSUMPTION
(Thousand Short Tons Contained Manganese)

Form and Use	Actual 1981	Projected 1995	Projected 2010
Ferroalloys and Metal	775.1	923	1,135
Carbon Steel	560.0	600	650
Stainless Steel	15.5	27	48
Other High-Strength Low-Alloy Steel	55.7	84	130
Alloy Steel	107.1	150	220
Other	36.8	56	87
Ore	159	215	310
Iron and Steel	71	85	100
Chemicals and Other	88	130	210
Total	934	1,138	1,445

SOURCE: Charles River Associates estimates.

used blast furnace technology. The economics for blast furnace ferromanganese production have deteriorated steadily, relative to electric furnace production. Second, the Johnstown, Pennsylvania, plant of Bethlehem Steel was damaged in a major flood in the 1970s and never reopened. Third, many less developed countries (such as India) have encouraged investment in downstream processing of manganese ore, through tax policies, bans on ore exports, or direct equity investments. Fourth, several foreign governments have been heavily involved in minerals research and development (particularly Mintek, formerly the National Institute for Metallurgy, in South Africa and the National Metallurgical Laboratories in India), and in investigating methods for reducing the costs of mining, handling, and processing manganese ores.

Most importantly, the costs of labor, energy, pollution control, and overseas transportation have risen rapidly in the United States, Europe, and Japan, compared with countries like South Africa. South Africa has low-paid labor, cheap coal, and access to inexpensive power based on coal. In addition, fuel cost increases over the last 15 years have driven up the cost of ocean shipping for both manganese ore and ferromanganese. Because both products are shipped in similar vessels, their ocean transportation costs are similar on a gross weight basis. Because manganese ore has a much lower manganese content than does ferromanganese, shipping ore to the United States instead of ferromanganese introduces a cost penalty equal to about 10 percent of the current price of imported ferromanganese.

In short, ferromanganese producers in the United States have trouble competing on a cost basis with modern, large, efficient producers in South Africa, and they have trouble competing on a political basis with producers in less developed countries that encourage ferromanganese exports over ore exports.

The United States could implement policies to improve the financial health of the domestic ferromanganese industry, both in the short run and the long run. Unfortunately, most of the short-run policies have major drawbacks.

In the short run the United States can attempt to help the ferromanganese industry by increasing its revenues or by decreasing its costs.

For example, severe tariffs, quotas, or floor prices for imports of ferromanganese would, at least in the short run, increase the prices received by the domestic ferromanganese industry. These higher prices would be passed on to the steel industry, the users of the ferromanganese. The ferromanganese producers would certainly benefit in the short run from the increased revenues realized through the higher prices, and the costs of the policy would not appear as a line item in the U.S. budget. Unfortunately, such a policy would have major problems. Most importantly, the precarious health of the U.S. steel industry makes it ill-equipped to absorb higher raw materials costs. It would not be able to pass on the higher ferromanganese

costs to its customers, because of the threat of steel imports. In theory, a companion tariff/quota/price policy could be imposed on steel imports, to reflect the higher manganese costs incurred by domestic producers. This would effectively pass on the costs to domestic consumers of steel, such as the auto industry, which in turn is also ill-equipped to absorb higher materials costs. Trade policies to support the ferromanganese industry would probably meet international opposition, directly from affected countries or through GATT.

Another method for raising revenues would be to subsidize the industry directly. Direct subsidies always face political problems. In addition, formulating subsidy policies to maintain domestic capacity in the face of international competition is a major problem. Consider the U.S. shipping industry. Coastwise shipping is protected by the Jones Act, which requires that all cargo carried between U.S. ports travel in U.S. flag vessels. International shipping has had no such protection. In order to maintain U.S. capacity to build ships and transport cargoes internationally, the United States for many years operated a construction differential subsidy and an operating differential subsidy. The government estimated the difference between U.S. and foreign costs for shipbuilding and international shipping, and gave the differential to the U.S industry. Unfortunately, such a program gives the U.S. industry no incentives to cut costs. The practical problems of organizing a subsidy program to maintain domestic ferromanganese production capacity should not be underestimated.

The United States could increase the revenues of the domestic industry in the short run by contracting with the industry to upgrade the strategic stockpile of manganese ore to ferromanganese. Such an upgrade seems quite sensible, anyway. If the United States did not have a domestic ferromanganese industry, a stockpile of manganese ore might be of little use during a supply crisis. Ferromanganese would be much more useful. Having the domestic industry upgrade the stockpile is a good way to maintain domestic capacity in the short run while guarding against its eventual loss in the longer run. Unfortunately, this policy works only as long as ore remains in the stockpile to be upgraded. After all the ore has been processed into ferromanganese, the policy will have run its course. This is a temporary but still useful solution to a long-run problem, and is currently being used for chromium.

One general way to reduce industry costs is through tax policies, but they seem inapplicable to the ferromanganese industry. Accelerated depreciation writeoffs, investment tax credits, and lower tax rates can encourage domestic production only if companies have operating profits to be taxed. Unfortunately, the domestic ferromanganese industry is generally incurring operating losses. Unless that situation changes, tax policies will not help.

Tax policies could be useful to multidivisional firms that make profits in other divisions, and they could be generally useful if tax benefits could be sold to other firms.

Government funding for research and development offers more hope, but such policies are generally effective only in the long run. They seem unlikely to help current producers of ferromanganese.

Research on new ways to make standard ferromanganese from traditional ores is unlikely to help the industry even in the long run. If new processing methods are developed for existing ores, these methods could be adopted in the countries in which the ores are found, still leaving the U.S. industry at a competitive disadvantage. The transportation cost disadvantages of U.S. ferroalloy producers might not even be eliminated, if domestic mine production were to take place in areas far from the ferroalloy producers and consumers.

The United States may seek an alternative by support for the development of the capability to make alternative manganese products that can be used by the steel industry. These manganese products, which could include electrolytic manganese or nonstandard grades of ferromanganese, may not be currently used by the steel industry, or may not be made by current producers.

Methods of handling new types of orebodies are potentially attractive. For example, research into collection and processing methods for Pacific and Blake Plateau manganese nodules could have large benefits in the long run, perhaps in 15 to 20 years. The Blake Plateau nodules have a relatively low manganese content (only 10 to 15 percent, compared with 25 to 30 percent for Pacific nodules), but their location in the U.S. economic zone off the coast of Florida makes them interesting as the highest grade domestic deposit. Research into the exploitation of new types of orebodies should go beyond ways to produce rather low-grade concentrates from low-grade ores mined in traditional ways. Manganese mining costs from existing mines are rather low per ton of manganese mined. Standard mining methods for low-grade ores combined with novel concentrating methods are unlikely to compete with standard mining and concentrating methods for South African ores, if standard ferromanganese products are to be made from both concentrates. Thought might be given to novel mining methods, such as solution mining, or to novel processing methods for low-grade ores.

3

CHROMIUM

INDUSTRY STRUCTURE

ORE PRODUCTION

The largest noncommunist producers of chromite are South Africa, Zimbabwe, Turkey, the Philippines, India, Brazil, and Finland. (Table 3-1). Smaller but important producers include Madagascar, Iran, and Greece. Since chromite content is highly variable from one producing country to the next, the production expressed as chromium content is included in Table 3-2. Essentially no chromite has been mined in the United States since 1961. The Soviet Union and Albania are by far the largest producers in the communist world and are currently the world's second- and third-largest ore producers overall. Considerable quantities of chromium flow from East to West, with Albania having become an increasingly important source of supply in recent years.

According to U.S. Bureau of Mines estimates, 98 percent of all known reserves occur in South Africa and Zimbabwe; according to IISI estimates, only 88 percent of reserves are located in these countries. World reserves are considered sufficient to meet conceivable demand for many decades. Noncommunist reserves outside Zimbabwe and South Africa, however, could satisfy noncommunist demand for less than 15 years, should a major long-run disruption occur in those countries.

The initial development phase for a chromium mine may range from less than a year for relatively small surface mines in settled areas to several years for large underground mines in more remote areas. Expansions can sometimes require less than one year, particularly in South Africa. As most mines tend

Table 3-1

CHROMITE PRODUCTION BY MAJOR PRODUCING COUNTRIES
(Thousand Metric Tons Gross Weight)

Country	1970	1973	1974	1975	1976	1977	1978	1979	1980
Albania	468	611	715	779	830	880	989	1,016	1,080
Brazil	27	73	108	173	186	310	269	340	349
Finland	121	148	165	331	175	169	178	177	175
India	271	288	396	500	402	353	266	309	319
Iran	200	140	175	172	160	233	198	136	82
Madagascar	131*	158	156	194	221	165	138	128	150
Philippines	566	581	530	520	431	538	537	561	572
South Africa	1,427	1,649	1,877	2,076	2,409	3,059	3,144	3,297	3,415
Turkey	519	436	666	717	581	508	376	454	399
U.S.S.R.	1,751	1,905	1,950	2,077	2,087	2,177	2,313	2,404	2,449
Zimbabwe	363	544	590	590	864	678	478	542	554
Other	209	163	133	159	147	150	152	170	185
WORLD TOTAL	6,053	6,696	7,461	8,288	8,493	9,220	9,038	9,535	9,730

*Exports.

SOURCE: U.S. Bureau of Mines, Minerals Yearbook, various issues, Washington, D.C.: U.S. Government Printing Office.

Table 3-2

WORLD PRODUCTION OF CHROMIUM ORE: 1960-1979
(Thousands of Metric Tons of Cr Content)*

COUNTRY	1960	1970	1973	1974	1975	1976	1977	1978	1979
Brazil	2	11	35	27	54	60	95	84	93
Finland**	--	21	68	72	89	113	153	129	115
India	35	90	96	131	165	133	116	88	89
Iran	22	73	46	58	51	53	77	66	46
Madagascar	--	36	36	54	68	74	57	41	44
Philippines	172	136	145	132	130	108	135	134	140
South Africa	236	442	511	581	643	746	948	974	1,022
Turkey	154	169	125	161	219	206	198	143	170
Zimbabwe	199	176	276	246	306	302	237	167	189
Other	120	76	52	48	45	45	44	44	42
Western World	940	1,230	1,390	1,510	1,770	1,840	2,060	1,870	1,950
Albania	83	135	184	207	226	241	255	270	290
Soviet Union	320	612	665	682	728	735	763	805	840
World Total	1,350	1,970	2,240	2,390	2,730	2,820	3,080	2,950	3,090

*Estimated.

**Based on chromium content of run-of-mine ore.

SOURCE: International Iron and Steel Institute. 1981. Chromium and the Steel Industry. Committee on Raw Materials. Brussels. March.

to be small-tonnage operations, compared to iron and coal operations, they must be located near existing infrastructure. Approximately 75 percent of all ore is mined by underground methods. The particular techniques employed range from traditional hand methods at small operations in developing countries to sophisticated mechanized methods in the larger mines in more highly developed nations.

World supply of chromium has been more than adequate to meet demand throughout the 1970s, although transport bottlenecks have occasionally caused spot shortages. Total world production rose from 5,844 thousand metric tons in 1970 to 9,542 thousand metric tons in 1980. Several notable developments in the supply of chromium have occurred since the mid-1970s. Among the most important are the emergence of Albania and the re-emergence of Zimbabwe as important suppliers to the West, as well as a significant shift in ferrochromium production away from the advanced industrialized nations towards the major ore producers.

FERROCHROMIUM PRODUCTION

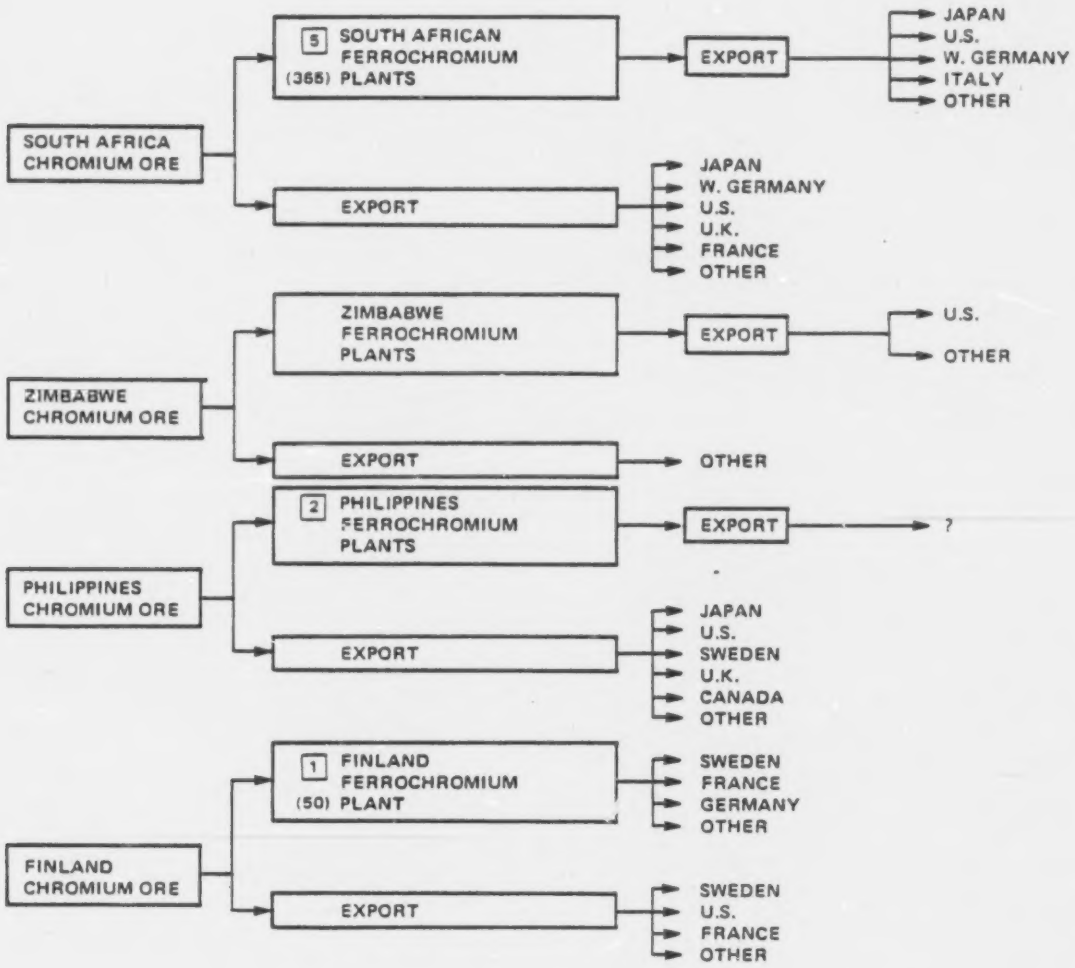
The flow of ore from the major producing countries in the western world to domestic or foreign ferrochrome plants and exports of ferrochrome from ore/ferrochrome producing countries are included in Figure 3-1. The world production of ferrochrome nearly doubled from 1973 to 1980 (Table 3-3). During this period, the production of the major ferrochrome, non-ore producing countries has declined only slightly. More pronounced has been the rapid growth of ferrochrome production in the ore-producing countries and the Soviet Union.

Recent advances in technology, combined with the ore producers' proximity to resources and favorable cost structures, have been the primary reasons behind this trend. National stimuli have been provided by increased demand from developing steel industries as well as foreign government policies aimed at increasing the value of exports and reducing reliance on imported ferroalloys. The major chromium-consuming countries have also suffered rising costs for energy, transportation, labor, and environmental protection. Many western producers have been forced to suspend operations or reduce capacity. Partial or complete shutdowns have occurred in the United States, France, and Sweden. In Europe the only producers able to break even appear to be the integrated producers, like Outokumpu Oy, and those with access to low-cost hydroelectric power.

A summary of world ferrochrome capacity by company, country, and alloy type is included in Table 3-4. This is not intended to be inclusive as several plants have been decommissioned and some new capacity has been added since this survey was completed. Turkey, India, the Philippines, Greece, Albania, Madagascar, and other countries are expanding or contemplating capacity additions. Some construction has been held up by weak demand worldwide for ferroalloys but may resume as the steel industry improves and capacity is decommissioned in Western Europe, Japan, and the United States.

Figure 3–1

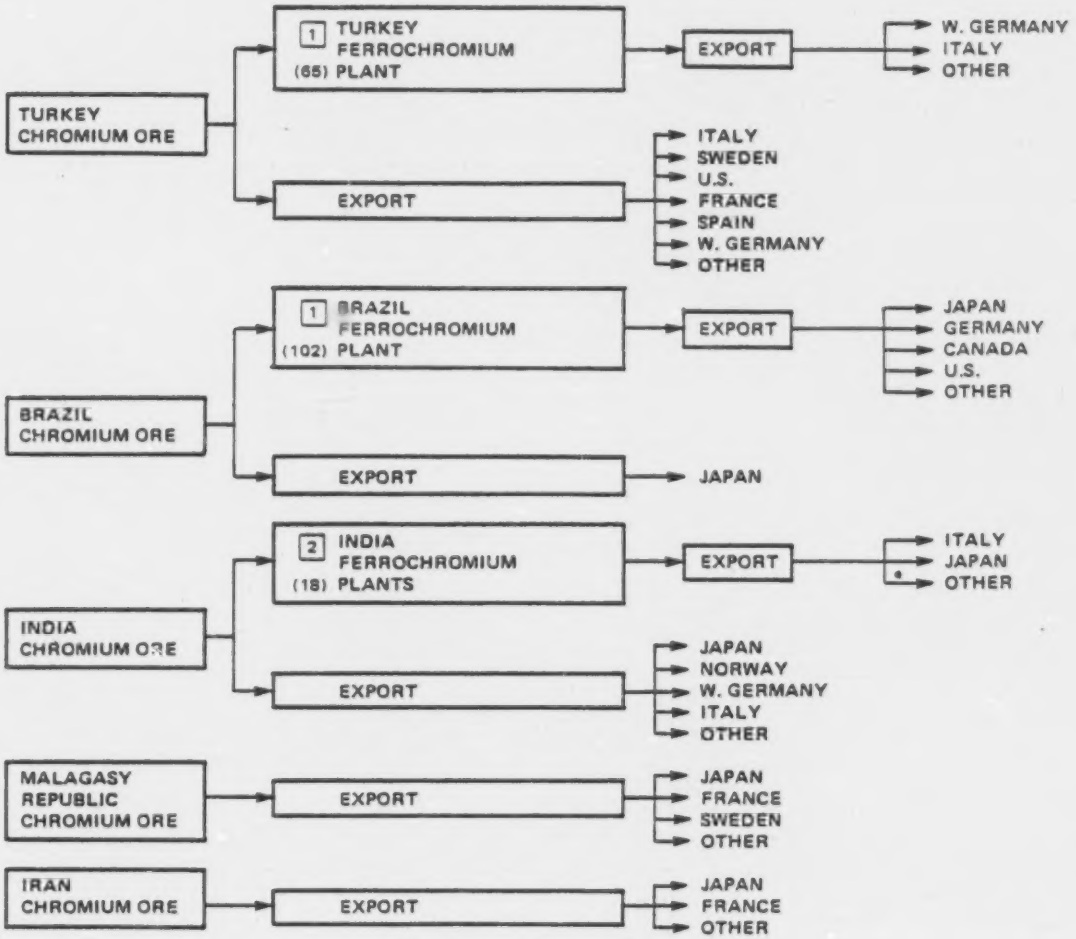
FLOW OF CHROMIUM ORE AND FERROCHROMIUM FROM THE MAJOR CHROMIUM ORE PRODUCING COUNTRIES



(continued)

Figure 3–1 (continued)

FLOW OF CHROMIUM ORE AND FERROCHROMIUM FROM THE MAJOR CHROMIUM ORE PRODUCING COUNTRIES (Continued)



□ Number of Plants ( ) Total Capacity (thousand metric tons of ferro-alloys)

SOURCE: Charles River Associates based on *Metal Bulletin Handbook*. Various issues. *Metal Bulletin Ferro-Alloys Survey*. 1979.

Table 3-3

FERROCHROME PRODUCTION BY PRINCIPAL COUNTRIES, 1971-1980
(Thousand Metric Tons Gross Weight)

Country	1971	1972	1973	1974	1975	1976	1977	1978	1979	1980
Brazil	10.2	11.6	15.7	38.1	52.6	65.3	66.2	62.6	84.4	93.4
Finland	35.3	24.3	39.6	48.1	39.9	39.9	33.6	44.5	49.0	49.9
France	90.2	94.5	138.5	111.6	99.8	101.6	101.6	92.5	95.3	86.2
India	12.3	1.5	6.6	15.4	10.0	17.2	18.1	21.8	22.7	16.3
Italy	36.3	41.6	37.0	39.9	44.5	45.4	39.9	37.2	42.6	40.8
Japan	403.1	302.6	494.8	541.6	486.2	463.6	399.2	274.0	381.9	427.3
Norway	36.5	30.1	30.1	30.8	27.2	31.8	22.7	15.4	11.8	11.8
South Africa	140.0	160.0	210.0	193.2	298.5	350.2	380.1	479.9	555.2	565.2
Spain	11.7	15.0	21.7	20.9	15.4	20.0	16.3	13.6	20.0	19.1
Sweden	63.8	58.3	80.4	100.7	92.5	116.1	134.3	166.0	189.6	188.7
United States	221.3	215.7	291.0	305.7	178.7	195.1	196.9	176.9	244.0	216.8
U.S.S.R.	109.0	120.0	120.0	184.1	205.9	480.8	635.0	662.3	689.5	698.5
Yugoslavia	18.5	27.1	27.5	39.0	53.5	42.6	36.3	51.7	65.3	64.4
Zimbabwe	60.0	70.0	85.0	181.4	181.4	186.0	199.6	199.6	199.6	199.6
Other	--	--	--	12.0	11.1	256.6	268.5	290.2	283.9	282.2
Total	248.2	1,172.3	1,565.7	1,862.5	1,797.2	2,412.2	2,548.3	2,588.2	2,934.8	2,960.2

Sources on following page.

Table 3-3 (Continued)

FERROCHROME PRODUCTION BY PRINCIPAL COUNTRIES, 1971-1980
(Thousand Metric Tons Gross Weight)

SOURCES: Roskill Information Services, Ltd. The Economics of Chromium.
U.S. Bureau of Mines. "Ferroalloys." In Minerals Yearbook. Washington, D.C.: U.S. Government Printing Office.

Table 3-4

WORLD FERROCHROMIUM CAPACITY, 1979
(Metric Tons Gross Weight)

	Charge	High-Carbon Ferrochromium	Medium-Carbon Ferrochromium	Low-Carbon Ferrochromium	Ferro-Silicon Chromium	Total All Grades
Brazil						
Ferbasa	90,000	--	--	12,000	--	
Canada						
Chromasco	50,000	--	--	--	--	
Finland						
Outokumpu	50,000	--	--	--	--	
France						
Sofrem	--	--	--	--	2,000	
West Germany						
Elektrowerk Weissweiler	*	*	*	*	35,000	200,000 (Incl. Charge)
India						
Ferro-Alloys Corp.	10,000	--	--	5,000	1,000	
Visvesvaraya Iron & Steel	*	*	*	*	--	1,560
Italy						
Lombarde Falck	--	--	--	15,000	--	
Montedison	--	40,000 (Incl. Charge)	--	--	--	

Table continued on following page.

Table 3-4 (Continued)

WORLD FERROCHROMIUM CAPACITY, 1979
(Metric Tons Gross Weight)

	Charge	High-Carbon Ferrochromium	Medium-Carbon Ferrochromium	Low-Carbon Ferrochromium	Ferro-Silicon Chromium	Total All Grades
Japan						
Awamura Metal Industry	--	72,000	--	--	5,000	
Japan Metals & Chemicals	--	147,200	--	42,800	24,400	
Nippon Denko	--	60,000		12,000	--	
Nippon Kokan	--	40,000	--	1,000	25,000	
Pacific Metals	--	24,900	--	--	--	
Showa Denko	172,700		--	62,500	27,000	
Mexico						
Minera Autlan	--	--	--	6,000	--	
Norway						
A/S Bjolvefossen	--	20,000	18,000	--	--	
Philippines						
Acoje Mining	--	*	--	--	--	
Ferro-Chemicals	--	*	*	*	--	2,000 (FeCr)
South Africa						
Feralloys	--	30,000	--	10,000	40,000	
Johannesburg Consolidated	120,000	--	--	--	--	
Palmiet Chrome	10,000	--	--	--	15,000	
RMB Alloys	20,000	--	--	--	--	
Tubatse Ferrochrome	120,000	--	--	--	--	

Table continued on following page.

Table 3-4 (Continued)

WORLD FERROCHROMIUM CAPACITY, 1979
(Metric Tons Gross Weight)

	Charge	High-Carbon Ferrochromium	Medium-Carbon Ferrochromium	Low-Carbon Ferrochromium	Ferro-Silicon Chromium	Total All Grades
Spain						
Ferroaleaciones y Electrometales	--	--	--	10,000	--	
Ferroaleaciones Espanolas	--	28,000	--	--	--	
Sweden						
Airco Alloys	--	180,000	--	--	20,000	
Ferrolegeringar Trollhatteverken	--	60,000	--	33,000	33,000	
Turkey						
Etibank	--	50,000	--	15,000	--	
United States						
Airco Alloys	--	70,000	--	--	30,000	
Chromium Mining & Smelting	--	30,000	--	--	--	
Globe Metallurgical Division	--	*	*	*	*	60,000
Namco	--	*	*	*	--	
Satralloy	--	36,000	--	36,000	23,000	
Tenn-Tex Alloy	--	*	*	*	--	30,000
Union Carbide	*	*	*	*	*	

Table continued on following page.

Table 3-4 (Continued)

WORLD FERROCHROMIUM CAPACITY (1979)
(Metric Tons Gross Weight)

	Charge	High-Carbon Ferrochromium	Medium-Carbon Ferrochromium	Low-Carbon Ferrochromium	Ferro-Silicon Chromium	Total All Grades
Yugoslavia						
Dalmacija	--	30,000	--	--	--	
Jugohrom	--	30,000	--	10,000	--	
Tovarna dusika Russe	--	8,000	--	5,000	5,500	

*Capacities not known.

SOURCE: Metal Bulletin Ferro-Alloys Survey. 1979.

Table 3-5 summarizes the recent status of world high-carbon ferrochrome capacity, the major segment of the market. Of the 2.5 million tons of Western world capacity, approximately one third was not operating. If the readily available capacity were reactivated, the effective capacity could be raised to about 80 percent of current design capacity, or about 2 million tons. The estimated practical capacity projections for 1985 reflect the likely decline in production in Western Europe, Japan and the United States, to be replaced by capacity expansion in the ore producing countries.

The low capacity utilization in ferrochrome production reflects the weak condition of the world steel industry, but the problems of many Western world ferrochrome producers reflect the high-cost structure both from a labor and energy-cost standpoint in many of the major producing countries. Also, technological changes have had a significant effect on the world steel industry. South Africa has been particularly innovative in producing ferrochrome from low-grade, friable ores. This product contains about 53 percent Cr but has been promoted to consumers extremely well by South African producers.

The U.S. domestic industry has been in dire straits since 1980, as evidenced by the cutbacks shown in Table 3-6. In 1981, domestic production supplied less than half of the U.S. requirements.

One significant difference between chromium and manganese for both ore and ferroalloys is East-West trade. The Soviet Union exports less chromium ore and ferroalloys to the West than it has in the past, but along with Albania, it is an important exporter. Unlike the case with manganese (Mexico and Brazil), no expanding resource bases are found in the Americas. Also, the capacity to produce ferroalloys in these western developed countries is declining. The source of ferrochrome raw material supply for the major countries is included in Table 3-7.

SOURCES

SOUTH AFRICA

Chrome ore is produced in two districts within the Bushveld Complex in the Transvaal: the Eastern Belt, or Lydenburg district, and the so-called Western Belt, or Rustenburg district. Essentially all South African ores are high-iron, containing about 44 to 46 percent Cr_2O_3, with Cr:Fe ratios between 1.3:1 and 2.4:1. Combined reserves in the Western and Eastern Belts of the Bushveld are estimated by South Africa at 1,063 million metric tons, to a vertical depth of 300 meters, with other resources about the same. The deposits are known to extend to well below 600 meters. Lesser reserves are found in the Potgietersrus and Zeerust areas, amounting to about 14.5 million metric tons. The upper group (or UG2) layer is characterized by Cr:Fe ratios of less than 1.4:1 and would not be mined exclusively for chromite; resources in UG2 are estimated by South Africa to total 630 million metric tons.

Table 3-5

WESTERN WORLD HIGH CARBON FERROCHROME PRODUCTION DATA
(Thousand Metric Tons Gross Weight)

	Total "Installed" Capacity 1980	Current Practical Capacity 1980	Estimated Actual Production 1980	Estimated Practical Capacity 1984-1985
Japan	450	380	350	320
Norway	25	10	5	0
United States	320	175	150	130
Spain	40	20	20	0
Zimbabwe	200	180	180	220
Sweden	280	150	135	135
South Africa	750	750	600	800
Turkey	50	40	30	80
Brazil	90	70	70	100
West Germany	100	40	20	0
France	60	40	40	0
Italy	40	30	30	0
Finland	65	60	55	60
India	NA	NA	NA	90
Philippines	NA	NA	NA	40
Other	40	20	20	50
Total	2,510	1,965	1,705	2,025

SOURCE: American Metal Market. 1981. Chromium Supplement. December 17.

Table 3-6

CURRENT STATUS OF U.S. FERROCHROMIUM PLANTS

Company	Plant Location	Status
Chromasco Ltd.	Woodstock, TN	Shut down
Elkem Metals Co.	Niagara Falls, NY, Marietta, OH, and Alloy, WV	Operating at low level
Interlake, Inc. Globe Metal Division	Beverly, OH	Operating intermittently
Macalloy Inc.	Charleston, SC	Shut down since December 1982
SKW Alloys, Inc.	Calvert City, KY	Shut down
Satralloy, Inc.	Steubenville, OH	Shut down since November 1982

SOURCE: U.S. Bureau of Mines and the Ferroalloys Association.

Table 3-7

PERCENT DERIVATION OF WORLD FERROCHROMIUM SUPPLY
FOR MAJOR WESTERN WORLD PRODUCERS, 1978

COUNTRY AND SUPPLY OF FERROCHROME (Thousands of Tons)		SOURCE OF CHROME ORE												FERROCHROME IMPORT SOURCES												
		S. AFRICA	PHILIPPINES	ZIMBABWE	TURKEY	BRAZIL	INDIA	IRAN	FINLAND	MADAGASCAR	OTHER	USSR	ALBANIA	S. AFRICA	SWEDEN	ZIMBABWE	YUGOSLAVIA	BRAZIL	NORWAY	FINLAND	INDIA	TURKEY	OTHER	U.S.	USSR	ITALY	SPAIN	FRANCE
549	South Africa	100																										
293	Japan	33	7		1	7	7	2		3	1	2		26	2			6			2		1					
199*	United States	28	8		3				3			8	2	36			7						5					
175	Sweden	5	8		9				48	3		9	4						13				1					
139	Zimbabwe			100																								
97	France	16			12			9		21	4	6			6				12	7			8					
67	Brazil		2			94					4																	
63	Yugoslavia										8	34	58															
53	West Germany	14	<1		<1			1	1		1	3	1	41	16			3		7		4		2	1	1	1	2
45	Finland							100																				
40	Turkey				100																							
37	Italy	7			12						5	1	4	23	21		12			1	2	4	4				3	
25	India						100																					

*Cr content.

SOURCE: Charles River Associates based on: International Iron and Steel Institute. 1981. Chromium and the Steel Industry. Committee on Raw Materials and Metal Bulletin Handbook, 1981.

The mines in both the Western and Eastern Belts are located 5 to 10 miles from railheads. The ore is transported by truck to the railhead. Once there, the ore must be moved to the heavily used port of Maputo (formerly Lourenco Marques) in Mozambique. Maputo is some 480 miles from the Western Belt and 350 miles from the Eastern Belt. Recently, a new mechanized operation was installed at the Matola berth to alleviate the perennial congestion at the port, which now can handle 2,500 metric tons of chrome ore per hour and can store up to 1.1 million tons. Possible alternate ports for use during political upheavals would be Durban and Richards Bay in South Africa, although a significant lead time would be required to accommodate chrome ore there.

Since chrome mines are relatively small (compared with manganese mines, for example), the expected expansion time for a mine is relatively short. In a typical South African underground chrome mine 50 percent expansion would require little more than one year. South Africa would be able to expand production quickly during a supply shortage. Transport bottlenecks would become the limiting factor, and a lead time of four years might be needed to double the port capacity. Short-run expansion plans in South Africa are ambitious, dictated mainly by what the market can bear. Long-run rates of expansion will be in line with total growth in world demand.

South Africa is currently the world's leading producer of both chromite and ferrochrome. It possesses approximately 70 percent of the world's chromium reserves. In 1979 there were 17 producing mines in operation, with an annual capacity of approximately 4.4 million metric tons. Production in 1980 was reported to be a record 3.4 million metric tons of ore and concentrate.

Some producers have begun to experience difficulties in the face of rising costs and stable prices. While most producers have maintained steady production, they are moving toward the rationalization of production from the least efficient mines, and closing many of the smaller mines. Producers running at close to full capacity throughout 1980 have accumulated large stocks.

South Africa's ferrochrome industry has experienced rapid growth since 1974. Between 1974 and 1980, South African production quadrupled, from 183,667 metric tons to 754,874 metric tons, while exports have tripled, from 179,512 metric tons to 596,847 metric tons. This has been reduced over the last three years due to the world recession. Low-priced South African exports have been highly successful in capturing significant market shares in Japan, the United States, and much of Europe. South Africa's ferrochrome boom (more than others) has been sparked by the new opportunities for Bushveld ore afforded by the AOD process and techniques for smelting friable ores.

South Africa enjoys the considerable advantages of a highly developed and sophisticated infrastructure, relatively low (although rising) transport and labor costs, relatively low-cost, coal-based energy, and virtually unlimited access to raw materials. Because ocean transport costs for chromium ore and

ferroalloys are substantial, amounting to about $35 per metric ton (gross weight) in 1981 for the South African-U.S. route, considerable savings in transportation costs can be realized by shipping ferroalloys (50 to 55 percent chromium) rather than ore (27 to 32 percent chromium). This factor alone gives South African ferrochrome producers a 3 to 5 percent cost advantage over those in the United States.*

Strong demand for ferrochromium in the mid-1970s sparked expansion of existing capacity and the addition of new facilities. Future expansion is not expected to be as rapid as that of the past decade, but is projected to occur on an incremental basis conforming to demand. Ferrochromium production is concentrated in six major plants, two of which produce low-carbon as well as high-carbon grades. Annual capacity is estimated at 800,000 tons per year.

Producers have consistently run at close to full capacity until recently, with a resulting accumulation of large stocks. Between 1975 and 1979, the production surplus varied between 20,000 and 70,000 tons per year, but in 1980 it shot up to over 155,000 tons. The cumulative surplus reportedly exceeds 375,000 tons, over 60 percent of 1980's total exports. Sluggish demand and low prices prompted the South Africans to reduce capacity utilization to between 50 and 60 percent in 1981.

Reflecting the growth of South Africa's domestic ferrochrome industry, the proportion of ore production that is exported has declined significantly, falling from 76 percent in 1969 to approximately 35 percent in 1980.

U.S.S.R.

The Soviet Union is the world's second largest producer of chromium ore. The two major ore producing regions are Kazakhstan and the Ural Mountain area. The Ural Mountain area is much less important as the chrome-to-iron ratio is low as well as the ore grade (<40 percent Cr_2O_3). The quality of the ore limits its use as a metallurgical product.

The vast majority of Soviet metallurgical ore for domestic consumption and the export market is located in Western Kazakhstan. High quality deposits such as Donskoye have a chromium-to-iron ratio of almost 4 to 1 with ore grades commonly exceeding 50 percent Cr_2O_3. Reserves have not been accurately reported but must exceed the U.S.B.M. estimates of 20 million tons by at least an order of magnitude given the capacity of new currently

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*Rail transport costs and port handling costs are more difficult to compare, because rail tariffs and port charges are not readily available. They should be similar per ton of materials, but U.S. rail tariffs are often based in part on the value of the commodity being shipped.

producing mines. The newest large Soviet deposit is the underground mine, Molodexhnaya, which was started in 1982 and has a capacity of 800,000 tons of crude ore. The weak demand in 1981 and 1982 for Soviet export ore has caused a reduction in Soviet mine output. The more aggressive stance by Albania in the export market has increasingly cut into Soviet exports.

The Soviet capacity for ferrochrome production is estimated at 420,000 tons of contained chromium, which is largely consumed domestically, although about 50,000 tons of ferrochrome is exported annually to Eastern Bloc countries and Western Europe. Not a great deal is known about the current state of Soviet ferrochrome operations, although the new expansion in ore facilities indicates likely expansion of ferrochrome capacity by the mid 1980s.

ZIMBABWE

All western trade sanctions against the importation of Rhodesian chromium were revoked in December 1979. Zimbabwe has moved back into European, American, and Japanese markets, although the emphasis has been on ferrochromium rather than chromite ore. The loss of skilled staff, poor markets, and labor disruptions resulted in reduced chrome ore output in 1982. The level of production was about 450,000 metric tons.

Rhodesian chrome has long been prized for its very high quality. Zimbabwe possesses the world's largest reserves of high-grade ore, with excellent potential for future discoveries. Although the development of the AOD process has significantly reduced demand for high-quality ore for metallurgical applications, Zimbabwe, with the world's second-largest reserves, remains important as a supplier of chromium. Although the movement away from high-quality chromite and ferrochromium production will reduce the demand for Zimbabwe's high-grade ores, it will permit the country to draw on large deposits of friable ore.

Most of the chrome lies in the Great Dyke, an elongated elevated geological structure that runs 300 miles or more in a northeast-southwest direction across the country. The ore is primarily stratiform, except for the Selukwe deposit, which is just off the southern part of the Dyke. Selukwe is a lode that produces some of the best ore in the world, typically containing 48 to 50 percent Cr_2O_3 with a Cr/Fe ratio of 3:1.

Transport problems have at times hampered Zimbabwe's ability to export chrome. In the past Rhodesia relied on rail transport to ports in Mozambique, principally Beira, which can handle 4,000 to 5,000 metric tons of ore per day. Port facilities at Beira do not compare with those at Maputo. The Rhodesia-Mozambique border was closed after Mozambique became independent in 1975; Rhodesian exports of chrome ore and ferroalloys thereafter had to move through South Africa. Urgent priority is being given to the restoration of the rail link to Beira and Maputo for the transport of bulk commodities. Annual production capacity of ore currently amounts to about 600,000 metric tons.

Over the last decade Zimbabwe has been making the transition from exporting chromite to exporting ferrochromium. Given the nature of its reserves, it has the potential to become the world's largest supplier of high-grade ferrochromium. In 1979, the two major producers, Rhodall and Rhomet, produced 235,000 tons of chromium alloys. The combined capacity of these two was 240,000 tons per year in 1980, and both have announced plans for important expansions.

Zimbabwe's ferrochromium has the potential to become very competitive internationally; access to ore is virtually unlimited, and costs, although higher than South Africa's, are low by world standards. Recent minimum wage regulations, loss of skilled personnel, rising electricity tariffs, and increasing rail costs may reduce this potential, however.

ALBANIA

Albania is the third-largest producer of chromite in the world, with 1980 production totaling about 1,080 thousand metric tons (run-of-mine-ore). Since the decline in Soviet exports of chromium ore and ferroalloys in 1979 and Albania's break in relations with China in 1978, Albania has turned to western markets to sell its chromium. Ore reserves and resources are not known with great certainty; reserve estimates range from 2 to 7 million metric tons of shipping-grade product, and resource estimates range from 15 to 17 million metric tons. These reserve estimates are almost certainly too low, because such reserves would not come close to supporting current and planned production levels. Other deposits are likely to be found, both because large areas have not been explored and because geological processes have created discontinuous, podiform deposits. Many deposits can be mined with open-pit methods.

Albania's last five-year plan called for 1,250 thousand metric tons of output in 1980, a goal that was not quite met. Five-year plans generally strive for ambitious growth rates, but usually fall short. Growth in exports is expected to be rapid through the mid-1980s, after which it should slow but still remain strong. Three ferrochromium plants completed in recent years have increased Albania's capacity to about 100,000 metric tons. Further expansion, possibly at the expense of ore exports, is quite likely.

Albania markets two grades of chromium ore: a concentrate with a 48 to 50 percent Cr_2O_3 content, and lumpy ore that contains 42 to 44 percent Cr_2O_3. Albanian exports have been criticized at times for not meeting their claimed standards. For example, 42 to 44 percent lumpy ore has been found, on delivery, to run as low as 38 percent Cr_2O_3.

TURKEY

With 1980 production of about 400 thousand metric tons of chromite ore, Turkey ranks as the sixth-largest producer of chromite in the world.

Turkey's reserves are somewhat limited and no real growth in production is expected over the next 20 years. Reserve estimates range from 5 to 20 million metric tons, while other resource estimates range from 10 to 35 million metric tons. Turkey will be forced to rely increasingly on lower-grade ores in the future, as reserves of high-grade ores are dwindling rapidly. The Cr:Fe ratio in the ores is relatively constant, however. Mining has shifted from open-pit to underground methods.

Short-run capacity expansion in Turkey is limited by the transport system. The main ports of Mersin and Iskenderun are 380 miles from the mineheads and have maximum loading capacities of 3,000 metric tons per day each. Limited capacity for rail and truck transport to the ports can constrain shipments of ore.

Exports of ore have declined in recent years as a result of increasing domestic consumption, general market oversupply, and Turkey's inability to meet price competition. A significant problem is the scattered nature of the Turkish podiform chromite ores, which limits the output and mechanization of many mines, and thus results in higher unit costs. Considerable stocks have accumulated. Ferrochrome capacity is estimated to be approximately 70,000 metric tons per year. An additional 100,000 tons per year is expected to be added by 1985.

PHILIPPINES

The Philippines is the leading producer of high-grade refractory chromite and an exporter of metallurgical-grade ore. Principal deposits are located in Zimbales province and Palawan Island. Plans for exploration and the establishment of new mines in other areas have been announced by several firms. The principal ore mined has a high alumina content, about 11.7 percent, a chromic oxide content of about 34.0 percent, and a Cr:Fe ratio of 2.4:1. While this ore is best suited for refractory uses, it can be blended for other applications. Output has been more or less constant over the past decade at 400,000 metric tons of refractory ore (34 percent Cr_2O_3) and 100,000 metric tons of metallurgical-grade concentrates (49 percent Cr_2O_3). Reports continually predict reserve depletion, but further exploration has always extended mining life by another 10 years. Current reserve estimates range from 3.0 to 10.0 million metric tons, with other resources ranging from 7.5 to 20.0 million metric tons. Gradual but constant growth in production may be expected for the rest of the century.

BRAZIL

Discrepancies exist in figures on Brazilian chrome ore production. The Brazilian Ministry of Mines reports a figure of about 600,000 metric tons of ore per year, but other sources indicate closer to 200,000 to 300,000 metric tons of ore and concentrate grading from 44 to 48 percent Cr_2O_3. Reserves are estimated to be between 2 and 9 million metric tons, with other resources

ranging from 20 to 31 million metric tons. Some ore is high-grade, but the average Cr:Fe ratio is only 2:1.

Most of Brazil's chromite is produced from mines in the Bahia area of northeast Brazil by Cia de Ferroligas S.A. (Ferbasa). According to U.S. Bureau of Mines figures, production has trended upwards since 1974 and Brazil has become a significant exporter of lump ore and concentrate.

Ferbasa is Brazil's only ferrochrome producer. Its facilities have been developed largely with Japanese assistance. Capacity for 1978 was given as 102,000 metric tons per year. Brazil is currently a substantial exporter of high-carbon ferrochrome.

INDIA

India is the seventh-largest chromite producer in the world, with annual output ranging between 300,000 and 500,000 metric tons of very high grade ores. Most production is carried out by a few companies mining in Orissa State. The Geological Survey of India has indicated recently that the proved reserve base is as large as 30 million metric tons in Orissa alone, up from the previous estimates of 5 to 6 million tons. Total resources in India may amount to as much as 90 million metric tons. Most of this ore is high-grade, above 48 percent Cr_2O_3, with a Cr:Fe ratio of 3.5:1. However, freight bottlenecks at Paradeep and other ports would limit the potential for rapid expansion during a world market disruption.

Production has been hindered at times in recent years because of power shortages in the producing regions. Ore exports have also declined recently, due to loss of Japanese business in response to price increases, and because domestic consumption for ferrochromium manufacture has increased. The Indian government has actively encouraged the development of the ferrochromium industry in order to increase the value of exports and to reduce dependence on imported ferroalloys. Capacity was estimated to be 262,000 metric tons per year in 1981, with expansion predicted for the next several years. Both Outokumpu Oy (Finland) and Showa Denko (Japan) have bid for the construction of modern charge ferrochromium facilities. Ferro Alloys Corporation is reported to have opened new charge chrome facilities near Bhadrak.

FINLAND

Finland's importance as a producer and exporter of chromite and ferrochrome has grown rapidly since 1972. The Kemi deposit of Outokumpu Oy produces all of Finland's chrome ore, with an annual output of approximately 800,000 metric tons grading 27 percent Cr_2O_3. The ore is subsequently concentrated to between 40 and 46 percent Cr_2O_3; capacity for salable products is about 400,000 metric tons. The Kemi deposit is variously reported to have 17 or 33 million tons of ore reserves and is located only 20 km from its port.

Another Finnish deposit that could come on-line in the future is in the Koitilaiskaira district; it has 47 million tons of 20 percent ore. The availability of cheap power and advanced ore-treating methods, in addition to the proximity to European markets, could conceivably make exploitation of this and similar deposits economically feasible. Total reserve and resource estimates vary widely, from 25 million metric tons of reserves and 50 million metric tons of other resources, to 50 million metric tons of reserves and 3 million metric tons of other resources. No real growth in production is expected, however, unless prices approximately double in real terms; in the long run, output could then increase by 25 percent. Over half of Finland's ore production is consumed by the domestic ferrochromium and steel industries.

While Finland's production is small relative to that of South Africa, the Soviet Union, and other leading producers, its geographical position and Western Europe's lack of ore enhance its importance somewhat.

MADAGASCAR

Madagascar is a significant producer of chromite, with annual output amounting to 150,000 metric tons of ore grading 49 percent Cr_2O_3. Significant growth is envisioned. A 50 percent expansion in production could be achieved during a disruption, given higher prices, because transport distances are short and port facilities are being modernized. The development of hydroelectric power may contribute to the development of a domestic ferrochromium industry.

UNITED STATES

Except for a small amount in 1976, the United States has produced no chromium ore since 1961. Chromium resources are estimated at 22.5 million tons of low-grade ore containing 1.8 million tons of recoverable chromium. These resources are located in Montana, Oregon, Washington, California, and Alaska. Mined and milled, these deposits collectively could yield 6.8 million tons of ore or concentrate averaging 38 percent Cr_2O_3.

The most promising deposits exist in the Stillwater complex in Montana, at the Mouat and Benbow mines (22.4 percent Cr_2O_3). Second in importance are the beach sands in Oregon (3 to 5 percent Cr_2O_3). Deposits in Alaska range from 15 to 45 percent Cr_2O_3, although CRA cost estimates indicate a very high cost of exploitation of even the most promising prospects. In California, nine areas have important chromite occurrences, ranging from 5 to 10 percent Cr_2O_3 up to 38 to 48 percent Cr_2O_3 in very small deposits.

Most significantly, the United States has experienced a sharp decline in its ferrochromium industry. Difficulty in competing with low-priced imports from the ore-producing countries had, by 1980, combined with depressed demand to force virtually all U.S. ferrochromium producers to suspend operations.

OTHER PRODUCERS

Other producers or potential producers include the Sudan, Cyprus, Greece, New Caledonia, Pakistan, Iran, and Colombia. These countries have modest reserves, although new exploration is being undertaken in several of them. The Japanese have undertaken feasibility studies for new production in the Sudan.

CHROMIUM METAL PRODUCTION

A small but important market exists for pure chromium metal in applications for which strength and corrosion resistance at high temperatures are critical. Chromium metal has been used in commercial applications since 1900. Publicly available data on chromium metal production, consumption, and stocks are scarce. The statistics presented here are CRA estimates obtained through industry contacts.

Production data for chromium metal are only available for the United States and Japan, as shown in Tables 3-8 and 3-9, respectively. By using U.S. consumption and industry stock data, CRA estimated net imports and production levels. U.S. production of chromium metal in 1980 was about 2,600 metric tons. Published data from Japan indicate that production in 1980 was at a level of about 3,500 metric tons, an increase of 23 percent from 1975.

CURRENT PROCESSING METHODS

FERROCHROMIUM

High-carbon ferrochromium is produced in a straightforward process similar to the production of electric furnace ferromanganese. Chromium ore, pellets, or briquettes are reduced with coke in an electric furnace in the presence of a blend of fluxes such as quartz, dolomite, and limestone. The alloy is generally 62 to 72 percent chromium, as determined by the chromium-iron ratio in the ore. The carbon content is typically 5 to 8 percent. Charge chrome made from South African ores usually grades 52 to 64 percent chromium. Such a low chromium content was not acceptable to consumers until the Argon-Oxygen Decarburization (AOD) process was developed for producing stainless steels.

One of the major producers of ferrochromium (charge chrome) in South Africa and in the world is Ferrometals Ltd., a subsidiary of Samancor. The Ferrometals Ltd. plant in the Eastern Transvaal near Witbank uses the submerged electric arc process to produce charge chrome. Chromium ore, having a chromium-to-iron ratio around 1.5 to 1.0 from Samancor's mines is mixed with quartz, dolomite, and coke breeze, and then fed to three open-hearth furnaces and to two closed-furnaces. Lumpy ore (containing from 39 to 41 percent Cr_2O_3) is smelted with briquettes (containing 38 to 42 percent Cr_2O_3) in a ratio of one to one in the two large furnaces. Ferrochrome from

Table 3-8

PRODUCTION AND CONSUMPTION OF CHROMIUM METAL
IN THE UNITED STATES, 1980
(Metric Tons)

	Metric Tons
Domestic Production*	2,576
Net Imports*	2,500
Stock Depletion	36
Consumption	5,112

*CRA estimates.

SOURCE: Estimates by Charles River Associates, February 1982, based on data obtained from industry contacts and U.S. Bureau of Mines, Minerals Yearbook, "Chromium," 1980.

Table 3-9

PRODUCTION AND CONSUMPTION OF CHROMIUM METAL IN JAPAN, 1975-1980
(Metric Tons)

	1975	1976	1977	1978	1979	1980*
Production	2,674	2,351	2,560	2,703	3,049	3,480
Total Sales	1,780	2,570	2,240	2,660	3,713	3,840
Consumption	850	1,100	840	1,010	1,253	1,350
Exports	930	1,470	1,400	1,650	2,460	2,490

*Preliminary estimates.

Note: Domestic sales are assumed to be equal to domestic consumption.

SOURCE: Roskill's Letter from Japan, various issues.

the electric arc furnaces is tapped off and a slag (containing 12 to 15 percent chromium) discarded. The charge chrome produced contains from 52 to 58 percent chromium and about 8 percent carbon. The briquettes used in ferrochrome production are made by blending and compacting the chromite fines with molasses, lime, and water. The process consists of drying the fines for about 45 seconds and then mixing with lime and molasses in two briquette mixers. The briquettes are made in a standard briquetting machine.

High-carbon ferrochromium can be used to produce very low-carbon ferrochromium (Simplex process) by combining it with a mixed non-chromium oxide and heating to about 1400°C in a vacuum. This process is now obsolete because of the low demand for the product occasioned by the AOD process for stainless steel. Simplex ferrochromium contains about 60 to 75 percent chromium and less than 0.02 percent carbon.

Another method used to produce relatively low-carbon ferrochromium (less than 0.01 percent carbon) is a combined carbothermic and silicothermic process outlined in the flowsheet shown in Figure 3-2. The consumption of both low-carbon ferrochromium and silicochromium has declined considerably since the AOD process was developed.

CHROMIUM METAL

Three commercial processes are currently being used to make chromium metal: aluminothermic process; electrolytic process; and direct carbon reduction (Chrome 98) process.

ALUMINOTHERMIC PROCESS. Producers of chromium metal by the aluminothermic process are widely distributed throughout the world and are found in the United Kingdom, the United States, West Germany, Japan, France, and Luxembourg as shown in Table 3-10. Total aluminothermic capacity in the noncommunist world is estimated to be between 8,200 and 9,100 metric tons per year, although capacity estimates can vary widely. Plant capacity is somewhat difficult to ascertain since the same equipment can be used for the exothermic production of alloys such as ferrovanadium, ferrocolumbium, or ferromolybdenum. Also, the ever increasing environmental restrictions are causing some limitations in the full utilization of installed capacity.

The aluminothermic process is a relatively simple one and is based on the reaction of high purity closely-sized aluminum powder and chromic oxide (Cr_2O_3), which are blended and charged to a refractory lined (usually Al_2O_3) reaction vessel (or converter). The charge is ignited, either by a potassium perchlorate/aluminum powder "wick" or by electrical means. The reduction takes place with the generation of considerable heat. The resultant high temperature (>2000°C) allows for a low-viscosity slag which separates readily from the metal. After the metal and slag is solidified and cooled, the chromium metal ingot is separated from the slag and then shot-blasted before crushing and sizing to specification.

Figure 3–2

FLOW SHEET FOR SILICOTHERMIC FERROCHROMIUM PRODUCTION

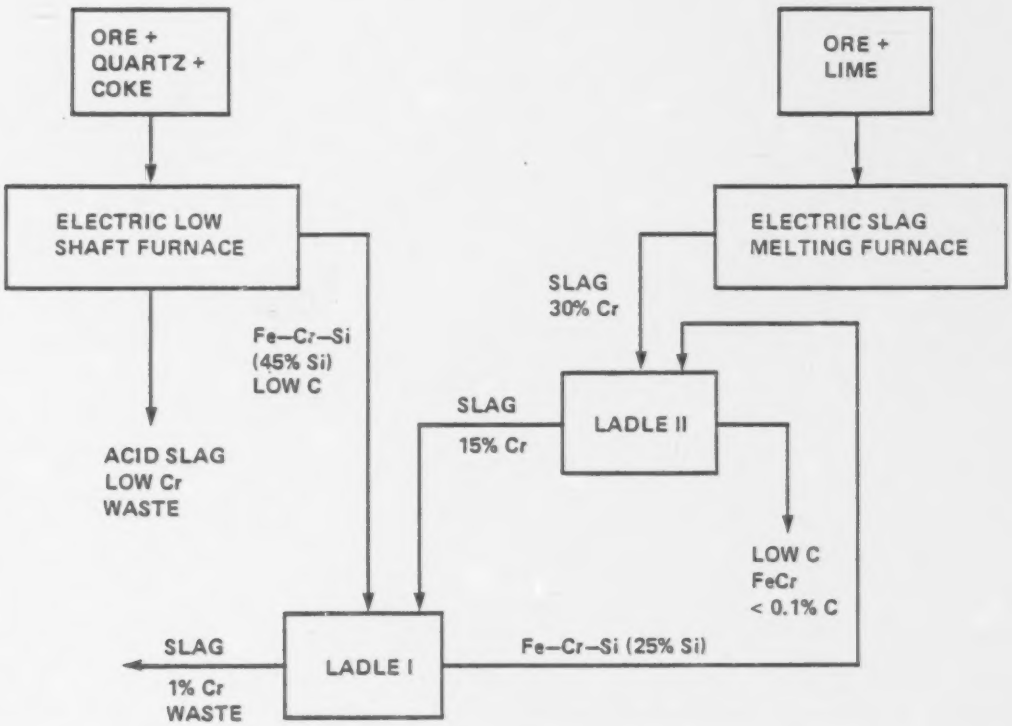


Table 3-10

PRODUCTION CAPACITY FOR ALUMINOTHERMIC CHROMIUM METAL
IN THE NONCOMMUNIST WORLD, 1981
(Metric Tons Per Year)

Country	CRA estimates	Roskill Estimates
UNITED STATES		
Shieldalloy Corporation	1,400-1,800	--
JAPAN		
Nippon Denko	1,000	300-400
UNITED KINGDOM		
London and Scandinavian Metalurgical Murex Metals & Alloys (Wales)	4,000	2,000-2,500
FRANCE		
Pechiney Ugine Kuhlmann	450	500
Delashore	450	500
WEST GERMANY*		
Goldschmidt	200-300	
Gesellschaft fur Electrometallurgie	400-900	500-700
LUXEMBOURG		
Continental Alloys	500	--
Total	8,200-9,100	3,800-4,600*

Notes on following page.

Table 3-10 (Continued)

PRODUCTION CAPACITY FOR ALUMINOTHERMIC CHROMIUM METAL
IN THE NONCOMMUNIST WORLD, 1981
(Metric Tons Per Year)

*There is one "captive" producer in Germany. It has substantial production, all of which is directly sold to two or three companies in Germany.

SOURCE: Estimates by Charles River Associates, based on data obtained from industry contacts, and Roskill's Letter from Japan, July 1981.

The purity of the final metal product depends mainly on the purity of the raw materials going in, especially the chromic oxide. There are two main grades of aluminothermic (AT) chromium. One is Chromium 99, suitable for air-melting applications (stainless steels, welding, and hard facing alloys), and the other is a high-purity grade called vacuum melting grade, which usually comes from selected lots with low residual contents of nitrogen and oxygen. Excess aluminum has been used in these lots to drive down the oxygen content. This high purity grade can partly replace the electrolytic chrome briquette in making superalloys for jet engines, but due to possible slag inclusions, this material does not go into the most critical hot-rotating parts of the jet engine. Instead, it is used in the combustion section of the engine.

High purity (metallurgical grade) chromic oxide is produced by roasting the chromite ore with alkalis like soda ash at red heat. This produces a slaggy clinker of sodium chromate which is then acidified to sodium dichromate by crystallization from the solution. The sodium dichromate is mixed with sulfur and calcined to produce a clinker. The soluble salts are washed, leaving behind the insoluble, purified chrome oxide. This process leaves a residual sulfur content that is both high and variable in the oxide.

British Chrome and Chemical Company in the United Kingdom has developed a proprietary process to make an oxide very low in sulfur and other metallic impurities. The company is the key supplier in the noncommunist world of chromic oxide to the aluminothermic chromium metal producers. British Chrome's annual capacity is 12,700 metric tons per year of chromic oxide. The company has built an oxide facility in the United States, called the American Chrome and Chemical in Corpus Christi, Texas. The plant produces sodium chromate and dichromate, some of which is used to produce chromic oxide for metallurgical and refractory applications. The capacity of the plant is 32,000 short tons per year of sodium dichromate.

Metallurgical-grade chromic oxide is also produced in West Germany on a small scale. Some oxide has been offered from the Soviet Union and Poland but little is known of the production capabilities of these two countries. The Soviet Union has recently withdrawn both its oxide and chrome metal from noncommunist markets. The United Kingdom is believed to have a total maximum converter capacity for AT chromium of 6,500 to 7,500 metric tons of metal per year, while continental Europe is believed to have a maximum of 5,000 metric tons capacity. However, this capacity is not entirely available, since other alloys are being processed with some of the same equipment.

In the United States, metallurgical-grade chromic oxide is produced by Shieldalloy Corporation, the country's sole AT chromium producer. Shieldalloy's proprietary process yields a low-sulfur, low-nitrogen chrome metal product. All of their oxide is consumed internally. Their capacity is roughly 1,400 metric tons per year of chrome metal. This could be expanded to 1,800 tons per year by bringing in some furnaces used for other alloys.

Separating the iron from the chromite in the ore to produce high-purity chromic oxide is relatively capital-intensive; converting the high-purity oxide to metal is not.

ELECTROLYTIC PROCESS. As shown in Table 3-11, electrolytic chrome metal comes chiefly from two noncommunist world plants of approximately equal size, one in the United States and the other in Japan. Elkem Metals Company (which purchased its plant in 1981 from Union Carbide) has plans to expand from 2,800 to 4,500 metric tons per year capacity within two years. Toyo Soda has a capacity of 4,000 metric tons per year.

These plants use processes based on U.S. Bureau of Mines developments of the 1950s and are fed by ferrochrome alloys of about 68 percent Cr grade. The ferrochrome as a feed is better and cleaner than chromite ore but chromite ore can be used. Elkem makes its Simplex ferrochrome at the same site as its electrolytic chrome.

In the electrolytic process the fine-ground ferrochrome alloy is dissolved in an acidic solution. To remove iron, ammonia is added to the solution which precipitates iron as ferrous ammonium sulfate. Addition of more ammonia precipitates chromium as pure chromium ammonium sulfate (chrome alum). The pure chrome alum is dissolved in water to provide the electrolyte feed for the cells. The cells operate continuously with chromium metal electrodepositing on the cathode. Periodically the cathodes are removed from the cells and replaced with undeposited ones. Chromium metal is washed and broken from the cathodes as 2- to 3-inch flakes about 1/16 to 1/8 inch thick for sale as "regular" grade or for further purification.

Electrolytic chromium is the chief grade of high quality metal required in the superalloys used in the hot, rotating parts of the jet engine, where the residual oxygen and nitrogen contents must be kept very low. For the high nickel superalloys (such as Hastelloys) used in incinerators, other chromium metal grades could be used, such as AT rather than electrolytic chrome metal.

CHROME 98 PROCESS. Besides electrolytic chrome, Elkem also produces Chrome 98 at their Marietta, Ohio plant. This is a metallic product made by reducing the oxide with carbon under vacuum. The chromic oxide and carbon are mixed together and compacted into briquettes. The briquettes are heated close to the melting point in a vacuum furnace for several days. The carbon is driven off as carbon monoxide, which leaves briquettes of pure chrome metal. This process is said to be similar to Union Carbide's (now Elkem's) Simplex low carbon ferrochrome process for removing the carbom from ferrochrome. The capacity for Chrome 98 can vary because the vacuum furnaces at Marietta can be used for making other products. Union Carbide had previously indicated that there are no capital installation costs associated with bringing Chrome 98 production on stream and that production could be

Table 3-11

PRODUCTION CAPACITY FOR ELECTROLYTIC CHROMIUM METAL, 1980
(Metric Tons Per Year)

Country	CRA estimates	Roskill estimates
United States		
Elkem Metals Company	2,800	3,000
Japan		
Toyo Soda Industry	4,000	3,000
TOTAL	6,800	6,000

SOURCE: Estimates by Charles River Associates, based on data obtained from industry contacts and Roskill's Letter from Japan, July 1981.

doubled easily with a small investment in additional mixing and briquetting equipment. The product can be used in place of electrolytic chrome where residual carbon and oxygen are not critical, such as in 75Cr-25Al master alloys and in welding rods (9 percent C). The annual capacity for this product is about 1,400 metric tons per year. Chrome 98 can be processed by vacuum melting methods to improve its quality to electrothermic grade chromium. However, the vacuum melting process is relatively expensive and will yield a product that may not be cost competitive with existing electrolytic chromium.

ALTERNATE PROCESSING METHODS

FERROCHROMIUM

A major new development in the ferrochromium industry is the use of plasma furnaces. As discussed in the section on alternate processing methods for ferromanganese production, the plasma furnace is capable of handling fines while also producing a high quality product. Middelburg Steel & Alloys in South Africa has been investigating the use of plasma furnace technology for the past four years, and is installing a 20MVA DC plasma arc furnace at the company's ferro-alloys plant at Krugersdorp (Figure 3-3). This is the first commercial application of this plasma technology in the ferro-alloys field. If the plant proves successful, larger scale plants for various processing applications will be available under licenses by Foster Wheeler Corporation of Livingston, New Jersey. Recent estimates by Foster Wheeler have shown that the capital cost of a plasma smelter producing 50,000 tons a year of high carbon ferrochrome, including a prereduction rotary kiln, would be about $15 million compared with $25 million for a conventional electric arc furnace. Operating costs are expected to be about 25 percent lower than in a conventional process. According to a Middelburg official, within two years after the furnace is installed the company hopes to have the system operating at peak efficiency and once the full system is evaluated the company will convert its other furnaces to DC operation.

CHROMIUM METAL

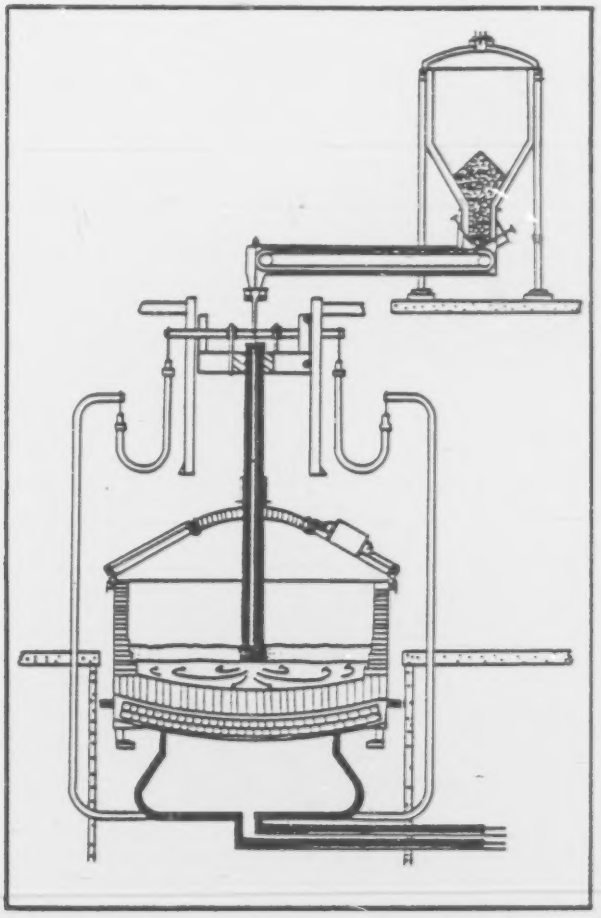
There are no new developments in the production technology for chromium metal. Refinements are being made to improve the yield in the electrolytic cells. In the aluminothermic process, the main efforts have been to upgrade further the quality and purity of the chromic oxide feed material.

DEMAND PROJECTIONS

As with manganese, chromium demand is primarily determined by trends in the steel industry. These have been discussed in the manganese section.

Figure 3–3

DIAGRAM OF MIDDELBURG STEEL AND ALLOYS' DC ARC FURNACE AT KRUGERSDORP, SOUTH AFRICA



SOURCE: *Metal Bulletin Monthly*. 1983. "Middelburg Strives to Stay Ahead in Charge Chrome." July.

Separate demand projections are provided (Table 3-12) for chromium ferroalloys (ferrochromium and ferrochromium-silicon), chromium metal (aluminothermic and electrolytic), and other chromium products (type not specified in data from the U.S. Bureau of Mines). The demand for superalloys is expected to grow about 3 percent annually from the base year of 1981 to 1995, and about 4 percent annually after 1995. The growth rate is expected to be lower in the earlier period because the aviation and aerospace industries will continue to face financial problems for the next several years. The consumption of chromium ore in refractories is expected to grow relatively slowly, about 2 percent annually, due to a continued phasing out of open hearth steel furnaces. It could even decline, if water-cooling becomes the norm for electric furnace steel production. Chromium ore demand for chemicals is expected to grow at about the same rate as industrial production.

FACTORS AND POLICIES AFFECTING U.S. PROCESSING

The factors that have contributed to the decline of the U.S. ferrochromium industry are essentially identical to those that have affected ferromanganese. The only important additional factor has been the Argon Oxygen Decarburization process for producing stainless steel, which permitted the low-chromium ores of South Africa to be used in producing ferrochromium.

The policy problems are essentially the same. Trade policies will damage the U.S. special steel industry, unless companion policies are instituted for imports of special steels. Such policies would impose costs on domestic consumers of special steels and raise objections from trade partners and GATT. Direct subsidization leads to political problems and practical problems of implementation. Upgrading of the stockpile is an excellent short-run expedient and is currently under way, but it does not solve long-run problems. Tax policies do not help an industry with no profits to tax, unless tax benefits can be sold to other firms or transferred to profitable operations of a ferroalloy producer.

Again, funding for research and development seems most attractive, which will only help domestic processing capabilities in the long run. As with manganese, research into methods for producing standard products from conventional ores does not appear helpful. An exception may be methods of producing high-quality chromium, such as electrolytic chromium and aluminothermic chromium. These products have high unit values compared with the large-tonnage chromium ferroalloys, and are needed for some of the most critical uses of chromium. Research into the exploitation of new types of orebodies, as with manganese, should go beyond ways to produce rather low-grade concentrates from low-grade ores mined in traditional ways. Standard ferrochromium products made from such concentrates could probably not compete with similar products made from South African ores. Unfortunately, no large resources like manganese nodules are available for chromium. The chromite

Table 3-12

PROJECTIONS OF U.S. CHROMIUM CONSUMPTION
(Thousand Short Tons Contained Chromium)

Form and Use	Actual 1981	Projected 1995	Projected 2010
Ferroalloys	246.2	406	701
Carbon Steel	5.2	5.6	6.1
Stainless Steel	174.6	300	540
Alloy Steel	52.4	79	120
Superalloys	4.3	6.5	11.6
Other	9.7	15	23
Metal	3.9	5.9	10.7
Superalloys	2.5	3.7	6.8
Other	1.4	2.2	3.9
Other Metallurgical Products	3.4	5.6	9.7
Total Metallurgical	253	417	721
Ore	106	154	229
Refractories	33	44	59
Chemicals	73	110	170
Total	359	571	950

SOURCE: Charles River Associates estimates.

contents of domestic laterites are rather low and the deposits rather small; revenues from a laterite processing operation in the United States would be dominated by nickel and cobalt. Nonetheless, they may be the best candidates for continued research into processing methods.

4

COBALT

INDUSTRY STRUCTURE

INTEGRATED PRODUCERS

The majority of cobalt production is from integrated producers in Africa. The source of the cobalt is the rich sedimentary copper deposits of the central African copper belt. The mineralized zone extends northwesterly from northern Zambia into the southern Zairean province of Shaba (Katanga) for a distance of approximately 300 miles. For both the integrated and non-integrated producers, cobalt is mined as a byproduct. Finland and Canada are the only other major countries with the capability of fully integrated production. Small quantities are also produced by fully integrated producers in South Africa. Estimated capacities for these companies appear in Table 4-1.

ZAIRE

The large production capacity and associated high cobalt grade of Zairean copper ore (0.3 percent Co) result in large quantities of cobalt reporting to the copper concentrates. In Zaire, where no cobalt concentrates are separately produced, there is little cost associated with cobalt production. Since Zairean copper is produced by direct electrowinning, all of the steps preceding the final cobalt extraction are necessary for copper production.

Although such high-grade sources of cobalt such as the Musunoi pit are being depleted, a strong reserve base of copper-cobalt ore still exists in Zaire

Table 4-1

COBALT PRODUCTION AND CAPACITY:
INTEGRATED PRODUCERS

Company	Mine	Refinery	1981 Production (Million Lbs.)	Nominal Capacity (Million Lbs./Yr)
Gecamines	Zaire (7)	Zaire (2)	24.5	37.0
ZCCM	Zambia (6)	Zambia (2)	5.7	11.0
Outokumpu	Finland (2) (Secondaries)	Finland	2.7	3.3
Falconbridge	Canada	Norway	3.2	4.0
Inco	Canada	England	0.9	2.0
Rustenburg	South Africa	South Africa	NA	1.1
			37.0	58.4

SOURCE: Charles River Associates, December 1983.

(Table 4-2). In the past few years, Gecamines (the state copper company) has ensured continued copper-cobalt production by construction of the huge KOV open pit mine complex and the DIMA concentrator in the western sector.

The flow of material in Zaire from mine to final metal product is fairly complicated so that a separate flow sheet is included (Figure 4-1). The majority of Zairean ore is concentrated and processed either at the older Shituru or the newer Luilu hydrometallurgical/electrowinning plants, both of which produce copper and cobalt. Some of the sulfide ore is roasted and subsequently treated at these copper plants. Some copper oxide and sulfide ore from both the western and central group mines is smelted at Lubumbashi along with the Kipushi mine output. No cobalt is produced at Lubumbashi.

The hydrometallurgical plants in Zaire precipitate cobalt hydroxide after copper has been stripped from the leached ores. The cobalt hydroxide is leached in acid and electrowon, to produce cobalt cathode. A "white metal" containing a copper-cobalt mixture is occasionally produced at the Panda electric furnace, in the central group near Shituru. In recent years, a considerable amount of the cobalt hydroxide has been stored and not further processed to avoid bringing too much cobalt on the market. The actual amount of cobalt held in stocks by Zaire and its sales agents as either metal or intermediate products exceeded 20,000 metric tons at the end of 1982.

The Zairean copper operations seek to maximize copper rather than cobalt recovery, so yields of cobalt from mine to metal in the 1970s were only in the 30 percent range. Most of the loss was in producing copper concentrate where almost 60 percent of the cobalt is lost in the tailings. Yields of cobalt from the copper concentrate are in the 70 to 80 percent range. At that point, cobalt production yields could be increased, but this is unlikely due to the current large stocks.

The cobalt cathode produced at Luilu is high quality, suitable for any purpose. The cobalt cathode produced at Shituru is low quality. This cathode is either melted and granulated in Zaire, or shipped to Belgium for processing into cobalt chemicals. When white alloy is produced at Panda, it is shipped to Belgium for refining. Metallurgie Hoboken Overpelt, a member of the Union Miniere/SGM/SGB group, operates the cobalt plant at Olen, Belgium.

ZAMBIA

Zambia produces ore from the same ore belt as Zaire, but the grades and forms of the cobalt ores differ significantly. The grade of copper is in the 2 to 4 percent range with cobalt averaging about 0.15 percent in the ores processed for cobalt recovery. Due to this lower grade and difficult mineralogical separation, it is impossible to get a high yield from the production of cobalt concentrates from the Zambian mines. The ores of Zambia are almost exclusively sulfide ores rather than oxide or mixed oxide-sulfide ores as in Zaire. This means that processing of copper is done by smelting

Table 4-2

COBALT RESERVES

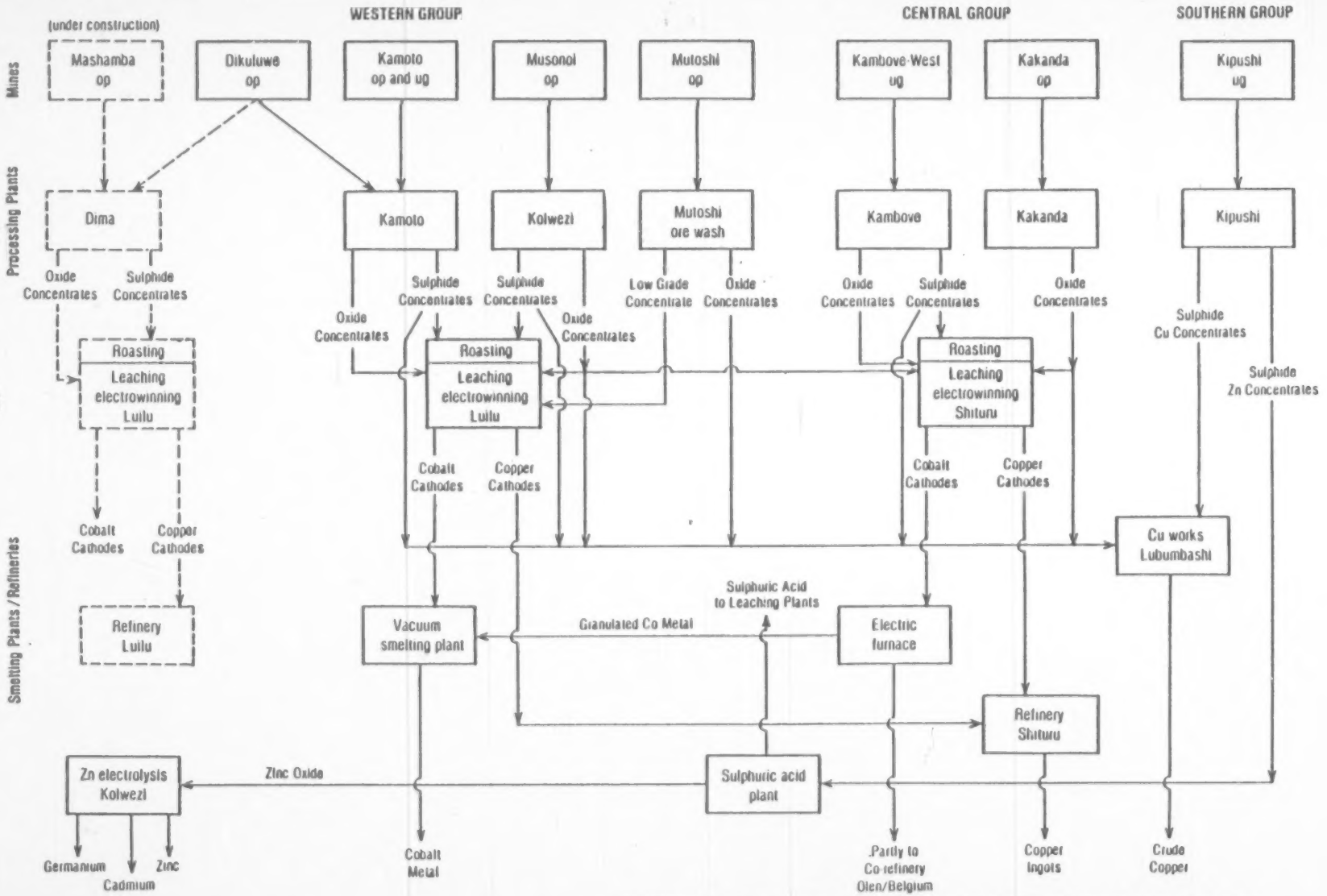
Country	Company	Mine	Type	Co (Mil. Lbs.)	Co (%)*	Other (%)*
Zaire	Gecamines	Musunoi	Cu-Co		1.4	5.21 Cu
		Mutoshi	Cu-Co		--	2.30 Cu
		Dikuluwe	Cu-Co		0.05	7.31 Cu
		Kamoto	Cu-Co		0.34	4.19 Cu
		Kakanda	Cu-Co		0.39	4.59 Cu
		Kambove	Cu-Co		0.35	3.95 Cu
				825 (Estimate)		

*Based on production grade.

SOURCE: Cobalt. A research report compiled in the German Federal Republic by the Federal Institute for Earth Sciences and Raw Materials, Hanover and the German Institute for Economic Research, Berlin, 1979.

Figure 4–1

MINES, DRESSING PLANTS, AND PROCESSING PLANTS OF GECAMINES IN ZAIRE



SOURCE: *Cobalt.* A research report compiled in the German Federal Republic by the Federal Institute for Earth Science and Raw Materials, Hanover, and the German Institute for Economic Research, Berlin, 1979.

rather than by hydrometallurgical extraction, so any cobalt reporting to the copper concentrate is lost to the copper slag. The cobalt split among the mill tailings, the cobalt concentrates, and the copper concentrates is about equally divided, resulting in only about a 33 percent recovery of cobalt into the cobalt concentrate. The yield is only about 75 percent in the cobalt refinery, for an overall recovery of only 25 percent.

Zambia Consolidated Copper Mines was formed in 1982 by the merger of Roan Consolidated Mines and Nchanga Consolidated Copper Mines. The former Roan Consolidated properties have a considerably higher recovery of cobalt in the cobalt concentrates (60 percent) with an overall yield of about 45 percent through the cobalt refinery stage. The higher grade Baluba and Chibuluma mines result in these better cobalt yields. Completion of new cobalt refinery facilities in Zambia and the consolidation of both producers to form ZCCM should streamline operations and increase overall cobalt yield. Current annual production capacity is 5,000 metric tons cobalt cathode, half of which is high purity.

FINLAND

Finland has a small integrated producer, Outokumpu Oy. The cobalt is derived principally from the Keretti and Vuonos, mines which are principally copper sulfide ores containing nickel, zinc, and cobalt. To avoid substantial cobalt losses a large amount of low-grade cobalt (0.7 percent Co) concentrate is produced. This is processed directly at the Kokkola metal refinery. The yield from mine to cobalt concentrate is only 65 percent and the refinery yield less than 70 percent from concentrate. Overall yield is about 45 percent.

Outokumpu supplements this cobalt with residues and processed nickel matte from the Harjavalata nickel plants. Nominal capacity is 1,500 metric tons of hydrogen-reduced powder annually. Production of cobalt salts was begun in recent years.

CANADA

Canada derives cobalt from the nickel mines owned by Inco and Falconbridge in the Sudbury District of Ontario, and from the Inco nickel mines in Manitoba. Falconbridge produces a nickel matte at its Sudbury, Ontario smelter that contains 47 percent Ni, 28 percent Cu, and 1 percent Co. Falconbridge ships the material to its Kristiansand refinery in Norway where a hydrochloric acid process is used to produce up to 1,800 metric tons of cobalt. Falconbridge also tolls some cobalt bearing material.

The feed for Inco's cobalt producing plant is obtained from the Sudbury area mines in Ontario and the Fort Thompson area mines in Manitoba. Inco processes these materials at their nickel refinery and produces a cobalt oxide. In the past this cobalt was sent to Inco's Clydach refinery in Wales

where it was simply ground and packaged or processed to salts. Inco has recently completed its cobalt refinery in Ontario to allow for complete domestic processing of its Eastern ores. This electrolytic plant at Port Colborne has a design capacity of 900 metric tons of cobalt annually.

SOUTH AFRICA

South Africa has recently joined the ranks of the integrated producers with installation of a cobalt circuit at the Johnson-Matthey nickel refinery by Rustenburg. Like the other South African platinum producers, Rustenburg derives cobalt from its nickel products separated during platinum ore beneficiation. The company formerly shipped about half its nickel matte to Amax at Port Nickel in Louisiana for recovery, and half to the Johnson-Matthey nickel plant, which previously did not recover nickel from the mattes. Capacity is approximately 1.1 million pounds of cobalt. The plant produces a cobalt sulfate.

NON-INTEGRATED SOURCES

AUSTRALIA

Australia produces intermediate cobalt products from both nickel sulfides and nickel laterites. Western Mining processes matte from its nickel smelter at the Kwinana refinery and utilizes an ammonia leach process to produce mixed nickel-cobalt sulfides. These are processed by Sherritt Gordon at its Canadian Fort Saskatchewan refinery.

The other major source of Australian nickel is Freeport's Greenvale mine. The ore is processed at the Yabulu hydrometallurgical plant. As at the Western Mining plant, the cobalt is removed utilizing an ammonia leach, and a mixed nickel-cobalt sulfide is produced. The mixed sulfide concentrate is shipped to the Nippon Mining refinery in Japan under a life-of-mine contract. Production of cobalt in sulfide at Greenvale reached 1,000 metric tons in the late 1970s. The Agnew project in Australia produces nickel sulfide concentrates that are smelted by Western Mining in Australia and refined by Amax in the United States. This project is a minor source of cobalt.

NEW CALEDONIA

Since most of New Caledonia's large nickel production is processed as ferronickel, most of the cobalt originating from the ores is not recoverable. The only significant production of cobalt from New Caledonian nickel ore is derived from the nickel matte produced by SLN. The New Sandouville refinery built to process this matte was partially destroyed by fire in 1979, but was quickly rebuilt. The plant produces chlorides by the solvent extraction process. The annual capacity is about 600 metric tons of cobalt in chloride solution. This solution was originally processed in Belgium by Metallurgie

Hoboken Overpelt, but it is now processed in France by Metaux Speciaux, a member of the Pechiney Ugine Kuhlmann group.

PHILIPPINES

The Philippines derives cobalt from its large lateritic nickel deposit at Surigao. The ore is processed at the Nonot Island nickel refinery. The ammonia leach process produces mixed nickel-copper sulfides. Sumitomo Metal Mining produces cobalt from these mixed sulfides under contract with Marinduque. Marinduque in the past announced plans to build its own cobalt refinery, but shortly thereafter renegotiated a new, 10-year contract with Sumitomo.

INDONESIA

Like New Caledonia, Indonesia processes most of its ore to ferronickel and therefore the cobalt is lost. Inco's Soroako mine ships some matte to the Clydach refinery in Wales to be processed to oxide or sales. The salt production at Clydach has been suspended since December 1982.

MOROCCO

The Moroccan production has become uneconomic due to the declining reserve base and increased mining costs. These are the only primary sources of cobalt in the world and await a better economic climate for possible reopening and a broadened reserve base.

OTHERS

Other minor sources of cobalt contained in ore include the Botswana RST Selebi-Phikwe nickel deposits, and South Africa's Western Platinum Mines.

NON-INTEGRATED REFINERS

All of the non-integrated refiners are listed in Table 4-3. Some of the cobalt sources listed in these charts fluctuate yearly for material that is not under long-term contract. The supply for AMAX and Sherritt Gordon, particularly, can vary from year to year. For instance, AMAX no longer obtains nickel matte from Rustenburg or SLN. Sherritt commonly processes the nickel concentrates from the Thompson, Manitoba mines of Inco, recovering cobalt from them. AMAX processes nickel matte from both Australia (Agnew) and Botswana. Both AMAX and Sherritt Gordon produce hydrogen-reduced powder. The cobalt powder produced by Amax is not acceptable for most metallurgical applications. In Japan, Sumitomo Metal Mining and Nippon Mining produced high-quality electrolytic cobalt. All of the sources for these cobalt

Table 4-3

COBALT PRODUCTION AND CAPACITY:
REFINERS

Company	Source	Refinery	1981 Production (Million Lbs.)	Nominal Capacity (Million Lbs./Yr)
Sumitomo	Philippines Indonesia Australia	Japan	5.3	3.7
Nippon Mining	Australia	Japan		2.9
Sherritt Gordon	Canada Australia S. Africa	Canada	2.0	2.2
AMAX	Botswana S. Africa Australia N. Caledonia	U.S.	1.0	1.0
Pechiney	Morocco	France	1.0*	3.3
W. Germany	Odds 'n Ends	W. Germany	0.9	1.1
			10.2	14.2

*Cobalt metal only.

SOURCE: Charles River Associates, December 1983.

producers are described in some detail in the preceding text and summarized in Figure 4-2. This figure summarizes the flow of material from mine to final product for all Western World producers. The production of cobalt is summarized in Table 4-4.

Since the large bulk of production is concentrated in Zaire and Zambia, it would be inappropriate to draw a flowsheet which includes inter-country flows.

CURRENT PROCESSING METHODS

COPPER SULFIDE ORES

The basic flowsheet for cobalt recovery in both Zaire and Zambia is shown in Figure 4-3. In this process the copper-cobalt sulfide ore is roasted and the calcine (together with oxide concentrates, in Zaire) is leached with sulfuric acid, and the copper removed by electrowinning. The spent electrolyte from the copper tankhouse is treated with a lime slurry to precipitate most of the iron and aluminum. Copper is precipitated in two stages, first at pH 5 and then at pH 6. The last traces of copper are removed by cementation on cobalt granules at pH 2.5. Cobalt is precipitated as cobalt hydroxide with additions of lime and is redissolved in spent cobalt electrolyte and sulfuric acid. Cobalt is deposited on stainless steel blanks during electrolysis, and is either marketed in granulated form or as broken cathodes. If the cathode in Zaire is of poor quality, the cobalt is refined in an electric furnace to remove sulfur, lead, and zinc, then granulated by pouring into water.

NICKEL SULFIDE ORES

There are three major processes used to recover cobalt from nickel sulfide ores (Figure 4-4). The first method is used by INCO in Thompson, Manitoba, by Matthey Rustenburg Refiners Ltd. (MRR) in South Africa, and by INCO at Copper Cliff, Ontario. Nickel concentrates are roasted in fluid bed reactors or multihearth furnaces and the roasted concentrates are smelted in electric furnaces or reverberating furnaces. The matte from the furnace is blown in converters, yielding a converted matte (50 percent nickel, 20 percent sulfur, and varying amounts of copper and cobalt depending upon the feed source). The converted matte is sent to a hydrometallurgical plant, where cobalt and nickel are separated and cobalt recovered as a powder, sulfate, chloride, or cathode.

The second method consists of drying the nickel concentrate to less than 1 percent water and injecting the concentrate into a flash smelting reactor. The matte from the reactor is blown in a converter and sent to a hydrometallurgical plant for nickel and cobalt recovery. This process is used by Outokumpu Oy at Harjavalta in Finland, by Western Mining (WMC) at Kalgoorlie in Australia, and by Bamangwato Concessions Ltd. (BCL) at Selebi Pikwe in Botswana.

Figure 4–2

FLOW OF COBALT FROM MINES TO FINAL PRODUCT

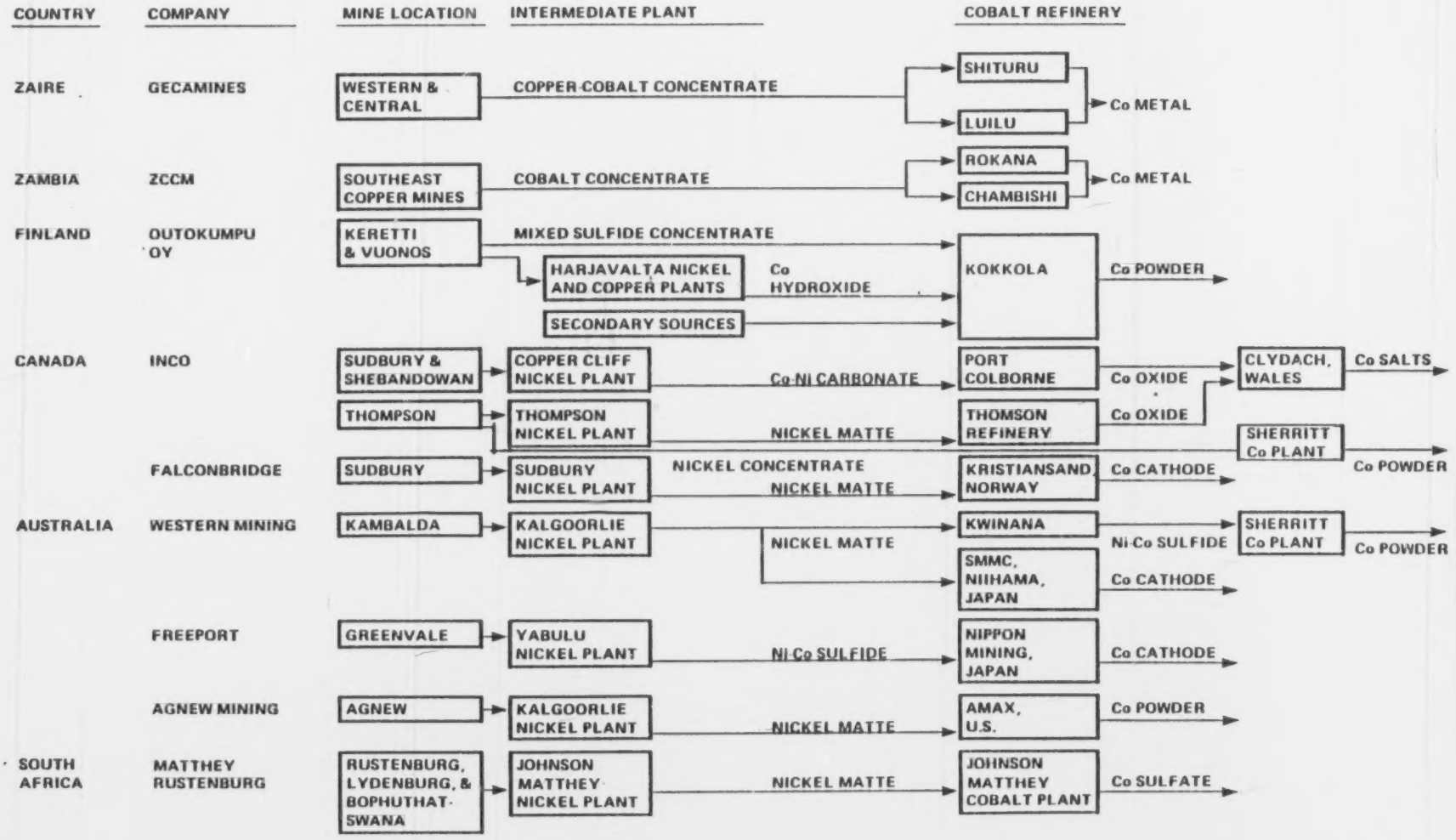
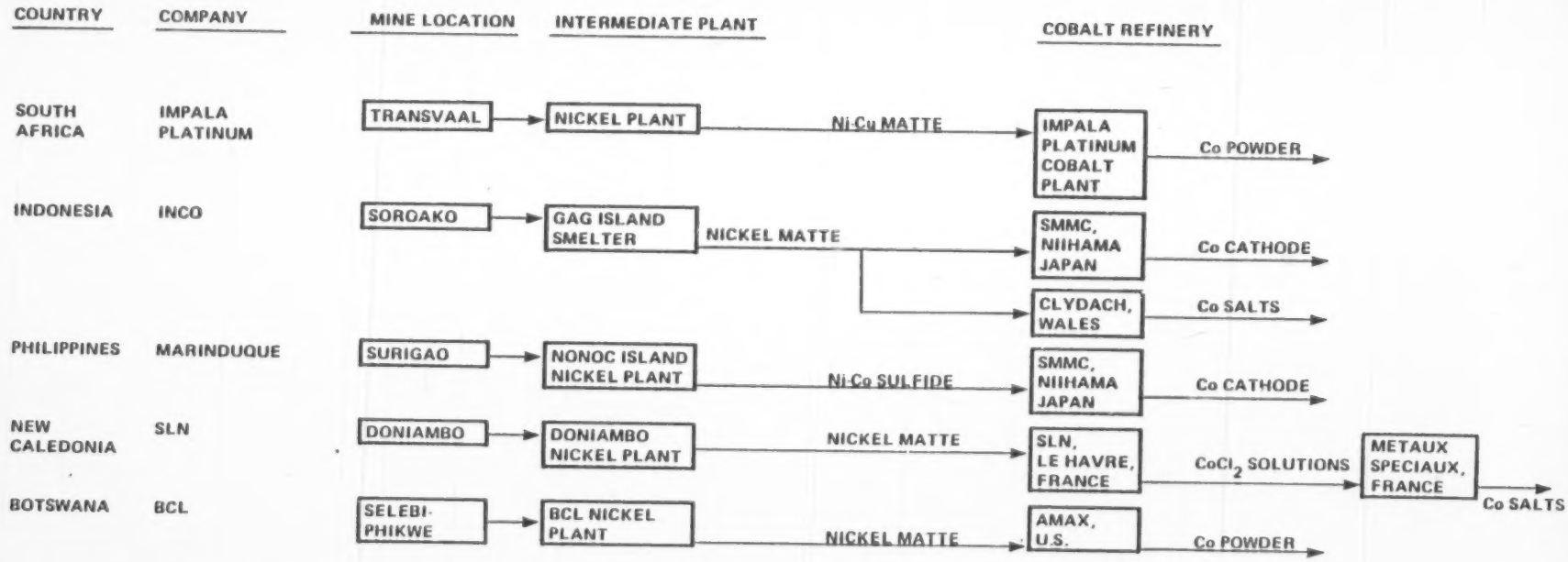


Figure 4–2 (continued)

FLOW OF COBALT FROM MINES TO FINAL PRODUCT



SOURCE: Charles River Associates, December 1983.

Table 4-4

PRODUCTION OF REFINED COBALT IN THE NONCOMMUNIST WORLD
(Metric Tons)

Year	Canada*	Finland**	France+	West Germany	Japan++	United States#	Zaire	Zambia	Total
1970	2,069	1,008	304	826	--	147	13,374	2,052	19,780
1971	1,961	925	576	601	--	140	14,518	2,080	20,801
1972	1,520	803	774	457	--	--	13,043	2,053	18,650
1973	1,517	1,010	835	370	11	--	15,052	2,664	21,459
1974	1,564	812	769	356	10	--	17,565	1,964	23,040
1975	1,490	820	769	340	48	30	13,638	1,843	18,978
1976	1,563	892	725	250	515	165	10,686	2,175	16,971
1977	1,890	985	852	400	1,093	221	10,215	1,704	17,360
1978	1,755	922	905	350	1,864	292	13,095	2,063	21,264
1979	1,862	1,162	769	385	2,653	421	14,029	3,176	24,475
1980	2,724	1,150	676	400	2,867	454	14,482	3,310	26,063
1981	2,785	1,230	447	400	2,421	406	11,124	2,570	21,383
1982[e]	1,176	1,460	500	400	1,940	454	5,800	2,446	14,176

*Cobalt refined by Canadian nickel companies in Canada, Norway, and Wales. Includes material from Canada, Australia, South Africa, and other sources.

**Primarily material of Finnish origin. Processing of East German residues began in 1979.

Footnotes and Sources continued on following page.

Table 4-4 (Continued)

PRODUCTION OF REFINED COBALT IN THE NONCOMMUNIST WORLD
(Metric Tons)

+Primarily material from Morocco. In 1979, processing of small quantities from New Caledonia began. Excludes production of oxides and sales.

++Material from Australia and the Philippines.

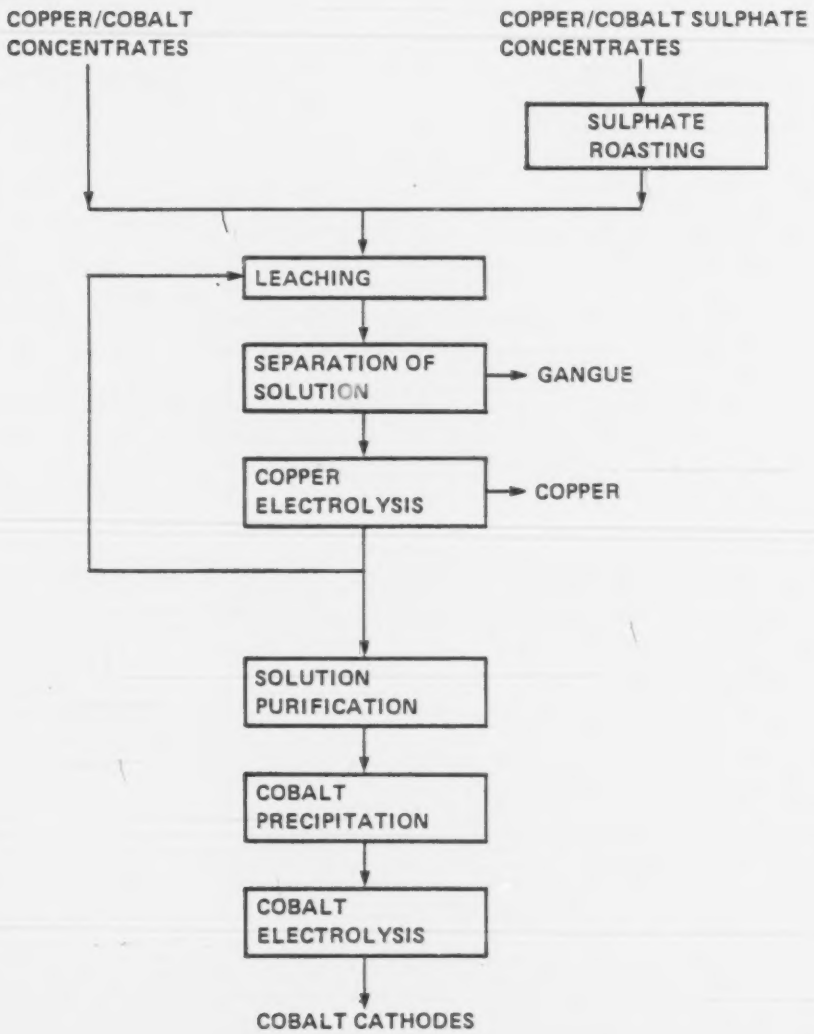
#Since 1975, includes material from Australia, Botswana, New Caledonia, and South Africa.

eEstimated.

SOURCES: U.S. Bureau of Mines. Minerals Yearbook (Annual). Washington, D.C.: U.S. Government Printing Office; contractor estimates; Company annual reports and 10-K reports. Copper Industry Service Bureau. Zambia Mining Yearbook (annual). Kitwe, Zambia.

Figure 4–3

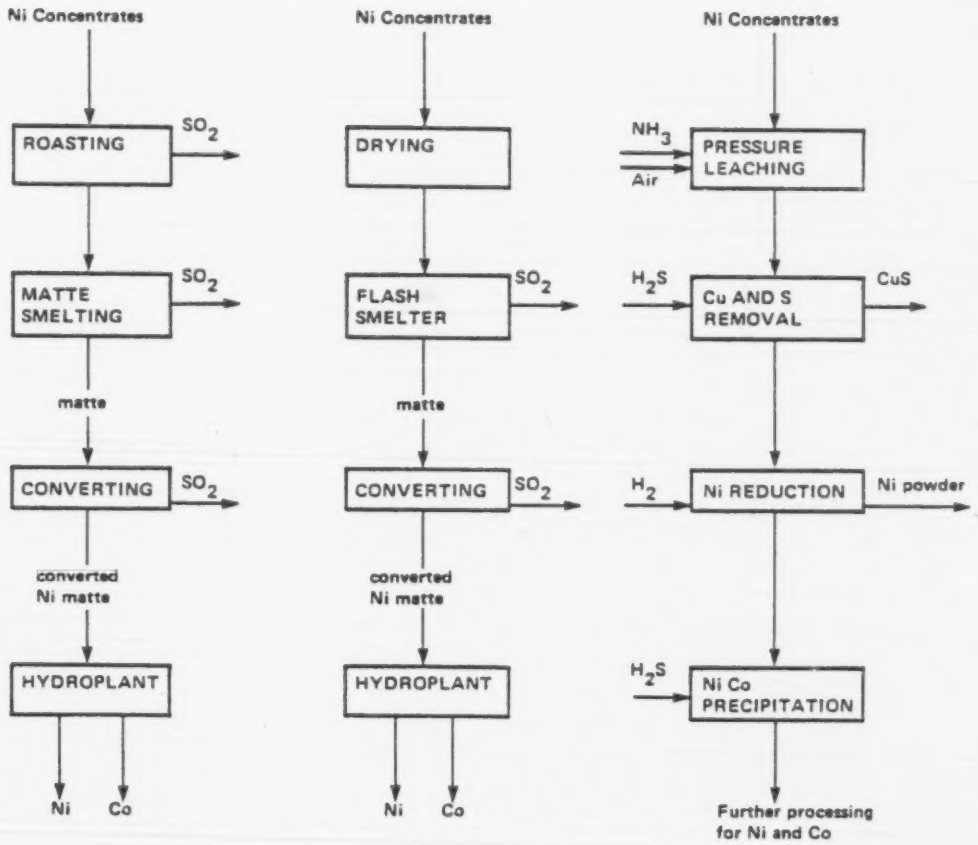
COBALT RECOVERY PROCESS FOR ZAIRIAN AND ZAMBIAN ORES



SOURCE: Brummitt, R. N. 1980. "An Review of Cobalt Production in Central Africa." Papers Presented at Canadian Institute of Mining and Metallurgy: 10th Annual Hydrometallurgical Meeting. Edmonton. October 26–28.

Figure 4–4

METHODS FOR PRODUCING COBALT FROM NICKEL SULFIDE CONCENTRATES



SOURCE: Lenoir, P., A. Van Peteghem, and C. Fenaau. 1981. "Extractive Metallurgy of Cobalt." In *Proceedings: International Conference on Cobalt: Metallurgy and Uses.* Volume 1. Brussels, Belgium. November 10–13.

The third method of recovering cobalt from nickel sulfide ores is the process used by Sheritt Gordon Mines Ltd. at their Fort Saskatchewan plant in Alberta. In this process both matte and sulfides are leached with air in autoclaves in ammoniacal media. After copper removal the solution is sent to an oxyhydrolysis step in which air is injected to convert the remaining sulfamates and thiosulfates to ammonium sulfate. The solution is then sent to a hydrogen reduction autoclave where nickel is precipitated as a powder. The resulting solution is treated with H_2S in an autoclave to precipitate mixed nickel-cobalt sulfides, which are then repulped and leached with sulfuric acid in an autoclave to remove iron. Cobalt solution is oxidized by air in the presence of ammonium hydroxide to form a soluble cobalt pentammine complex. Nickel is removed by adding sulfuric acid to form an insoluble sulphate nickel-ammonium salt which is recycled to the nickel refinery. The cobalt in solution is reduced by additions of cobalt metal powder and sulfuric acid and then precipitated as powder in a hydrogen autoclave.

The Sheritt process is also used by Impala Platinum in South Africa and by Western Mining Corporation's Kwinana Refinery near Perth, Australia.

The various ways cobalt is recovered are shown in Table 4-5.

NICKEL OXIDE ORES

Nickel oxide ores are either processed pyrometallurgically, into ferronickel or matte, or hydrometallurgically, yielding mixed nickel-cobalt sulfides.

MATTE PRODUCTION. To produce a nickel sulfide matte from an oxide ore, the oxide ore is first dried, calcined, and reduced in electric furnaces to ferronickel. Some of this ferronickel is blown in converters by injecting sulfur to form a nickel matte. The matte is further blown with air to oxidize most of the iron. Two companies have produced a matte in this manner: Societe Le Nickel (SLN) in their Doniambo plant in New Caledonia, and INCO in their Guatemala and Indonesia plants. The matte is processed in the same manner as nickel sulfide mattes in hydrometallurgical plants.

HYDROMETALLURGICAL PROCESSES. Nickel oxide ore at the Moa Bay plant in Cuba is processed using a sulfuric acid pressure leach. Nickel and cobalt are recovered as mixed sulfides. These mixed sulfides are sent to Eastern Europe for recovery of nickel and cobalt.

Hydrometallurgical processes are used to process laterite ores at two other operations: Queensland Nickel Pty. Ltd. in their Greenvale plant in Australia and the Marinduque Mining and Industrial Corporation (MMIC) in the Philippines. At the MMIC plant, a cobalt-nickel sulfide is produced and then shipped to Japan for refining by Sumitomo Metal Mining Co. (SMMC). At the Greenvale plant, a similar cobalt-nickel sulfide is produced and shipped to Japan for refining by Nippon Mining Co.

Table 4-5

PRODUCTION OF COBALT FROM NICKEL MATTE AND MIXED NICKEL-COBALT SULFIDES

Company	Leaching		Ni/Co Separation	Outlet Co	Capacity Metric Tons Co/Year
	Pressure	Media			
AMAX, Port Nickel	atm + pressure	H_2SO_4	AMAX Process	Powder	545
MRR, South Africa	yes	H_2SO_4	SX process	$CoSO_4$, $7H_2O$	100
NM, Hitachi	yes	H_2SO_4	SX process	Co cathodes	1,300
OUTOKUMPU, Kokkola	yes	H_2SO_4	SHERRITT process	Co powder	230
SLN, Le Havre	no	$FeCl_3 + Cl_2$	SX process	$CoCl_2$ solution	600
FALCONBRIDGE, Kristiansand	no	HCl Cl_2	SX process	Co cathodes	2,000
SMMC, Niihama	yes	H_2SO_4	SX process	Co cathodes	1,410
SHERRITT, Fort Saskatchewan	yes	NH_4O_4	SHERRITT process	Co powder	600

SOURCE: Lenoir, P., A. Van Peteghem, and C. Feneau. 1981. "Extractive Metallurgy of Cobalt." In Proceedings: Volume 1. International Conference on Cobalt: Metallurgy and Uses. Brussels. Belgium. November 10-13.

FINE POWDER PRODUCTION

Cobalt powder is produced by the reduction of the oxide, pyrolysis of organic salts, and hydrogen reduction of aqueous salt solutions under pressure. Reduction of the oxide yields a powder of 99.5 percent purity and with a grain size of about 1 micron, a powder used for the production of cemented carbides. The Outokumpu Kokkola plant in Finland produces extra-fine cobalt powder using this process. The capacity of the plant has been recently increased about 300 metric tons of extra-fine powder per year. In the process cobalt oxalate is thermally decomposed in a heat treatment furnace under a protective atmosphere.

DOMESTIC COBALT PRODUCTION

The AMAX plant at Port Nickel, Louisiana is the only domestic source of cobalt produced from imported cobalt-containing material. The feed to the plant consists of BCL matte from Botswana (42 percent nickel, 37 percent copper, and 0.5 percent cobalt) and Agnew Mining's nickel matte smelted at the Kalgoorlie nickel plant owned by Western Mining in Australia.

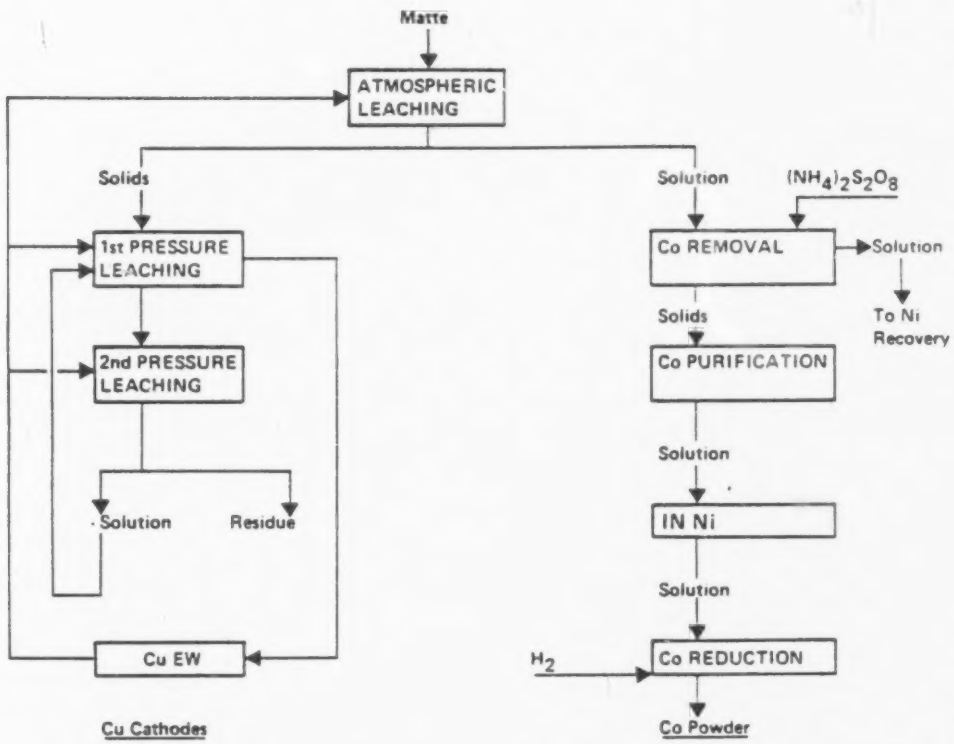
The mattes are crushed, wet-milled and blended. The major steps involved in processing the blended matte are shown in Figure 4-5. The matte is leached using the spent electrolytic from copper electrowinning. During leaching about half the nickel and cobalt contained in the matte go into solution, while the copper is cemented and the iron is kept insoluble by close pH control. Solids from this first leaching step are sent to a two-step pressure leaching process, and copper electrowon from the leach solution. The leach solution from the first leaching step is treated with ammonium persulfate to precipitate the cobalt. The cobalt-free solution is further processed for nickel recovery. The precipitate containing cobalt is leached with sulfuric acid and methanol in an autoclave. Nickel from the resulting solution is removed by acidifying the solution with sulfuric acid to a pH of about 2.5 and then cooling to about 32°F to crystallize out the nickel as nickel-ammonia sulfate. The slurry is centrifuged, and recycled to the matte leaching circuit. The resulting solution containing the cobalt is ammoniated to a pH of 8. Nickel is further removed by ion exchange (IX) before the cobaltic pentammine solution is reduced with hydrogen to produced cobalt powder. The cobalt powder is dried under hydrogen and can be sintered to cobalt briquettes, also under hydrogen.

Carolmet, Inc., owned jointly by companies in Belgium and Zaire, operates an extra-fine cobalt plant in Laurinburg, North Carolina. The plant has a capacity of about 1,000 metric tons of cobalt powder annually, using cobalt cathodes from Zaire as process feed. The technology is the same as that used by Metallurgie Hoboken Overpelt at its plant in Olen, Belgium. Details are not available.

GTE Sylvania produces extra-fine cobalt powder from tungsten carbide scrap at its plant in Towanda, Pennsylvania. Details are not available, but the

Figure 4–5

AMAX PORT NICKEL REFINERY



SOURCE: Lenoir, P. A. Van Peteghem, and C. Feneau. 1981. "Extractive Metallurgy of Cobalt." In Proceedings: Volume 1. International Conference on Cobalt: Metallurgy and Uses. Brussels, Belgium. November 10-13.

company has been issued several patents on the subject in recent years. Plant capacity is probably about 500,000 pounds of cobalt powder annually. Metallurgical International produces pre-mixed tungsten carbide-cobalt powders from tungsten carbide scrap at a small plant in Tinton Falls, New Jersey.

ALTERNATE PROCESSING METHODS

Continued research is being performed on nickel and cobalt recovery from laterites. Two potential processes being investigated are the AMAX process and the UOP process.

The AMAX process is a versatile acid leach process that is applicable to a broad range of nickel oxide ores. The process will have high recoveries of nickel and cobalt, as well as low energy consumption. This will allow the processing of low grade limonitic and garnieritic ores that cannot be processed economically at this time by pyrometallurgical processing.
The UOP process is a unique modification of the reduction roast-ammonia leaching process for oxide silicate ores of nickel and cobalt. The original process is used in several plants around the world. The process improvements are such that nickel and cobalt extractions well above these experienced commercially to date are possible. Cobalt is separated by solvent extraction and recovered as cathode.

Significant research is being performed on the development of techniques for mining and processing cobalt-bearing manganese nodules from the ocean floor. Two major techniques for processing manganese nodules, hydrometallurgical and pyrometallurgical, are under investigation. The principal factors to be weighted are energy requirements and reagent costs, and the need to recover four versus three of the principal elements in the nodules. A discussion of the major processes is given in Appendix A.

Continued research is being performed on the technology for recycling cobalt from a variety of catalysts. It is believed that nearly all cobalt could be recovered from spent hydrotreating catalysts in the near future. So far only molybdenum and vanadium have been removed from these catalysts. Two companies in the United States have recently announced that they will build plants to recover cobalt and the other useful metals from these catalysts. Cobalt is already being recovered from some other catalysts, less important in magnitude.

Research into the recycling of cobalt containing superalloy scrap has been performed by Certified Alloy Products, Inc. in Long Beach, California. Under present technology (vacuum induction melting) engine parts that have been in contact with combustion products cannot be directly recycled. The new process involves segregating contaminated scrap into pedigreed alloy lots, then melting the scrap in an electric arc furnace with oxygen lancing. This oxidizes all the reactive elements, such as Al, Ti, Zr, and Hf, and purges any elements with low melting points. The resulting material is

reconstituted by vacuum induction melting and is refined back into usable superalloys. The process can be used to upgrade other forms of chromium. However, the high losses of chromium would make the process unattractive. In addition, electrolytic quality chromium could not be produced.

DEMAND PROJECTIONS

Cobalt demand in superalloys will fall relative to the total demand for superalloys, due to a gradual movement toward nickel-base alloys that can tolerate higher operating temperatures in gas turbines (Table 4-6). We expect cobalt demand in superalloys to grow at an average rate of about 2 percent annually from 1981 to 1995, and 3 percent from 1995 to 2010. Cobalt demand in other uses will grow at about the same rate as industrial production in the long run, about 3 percent annually. Growth in the short run may be somewhat different, due to the lingering effects of the high cobalt prices in 1978-1979.

FACTORS AND POLICIES AFFECTING U.S. PROCESSING

The situation with cobalt is very different from that for the bulk ferroalloys of chromium and manganese. The United States has exactly two cobalt plants that do not depend on scrap as their feed: the AMAX nickel refinery at Port Nickel, Lousiana, and the Carolmet extra-fine powder plant in Laurinburg, North Carolina. Both plants are likely to continue operating into the foreseeable future, although the AMAX plant will suffer from the usual cycles in the nickel market. No policies should be needed to promote continued operation of these plants.

The Carolmet plant supplies a substantial fraction of U.S. requirements for extra-fine cobalt powder, and tungsten carbide recycling plants supply essentially all the rest that is needed. The situation with the Amax plant is quite different. Its cobalt capacity is about 1 million pounds annually, while U.S. consumption in the last five years has ranged from 10 to 20 million pounds annually. AMAX therefore supplies only 5 to 10 percent of total U.S. cobalt requirements, and all of its raw materials are imported.

Given the current weak state of the cobalt market, only direct subsidies would be likely to encourage the emergence of new capacity within the next five years. International trade in cobalt-containing intermediates is sizable, but current supplies generally move under long-run agreements or move from one processing plant to another within a single company. The chances are slim that any government policies could successfully encourage an increase in domestic processing of foreign cobalt intermediates, in either the short or long run.

Table 4-6

PROJECTIONS OF U.S. COBALT CONSUMPTION
(Million Pounds Contained Cobalt)

Use	Actual 1981	Projected 1995	Projected 2010
Carbides*	1.344	1.4	2.1
Superalloys**	6.285	8.3	12.9
Magnets+	1.687	4.3	6.9
Other metallurgical+	2.004	3.7	5.6
Chemicals++	3.961	5.4	8.5
Total	15.321	23.1	36.0

*Extra-fine powder.
**High-quality metallic cobalt, usually cathodes.
+Usually requires metallic cobalt.
++Can use metal, oxide, or other cobalt chemicals as feed for cobalt chemical production.

SOURCE: Data for 1981 from U.S. Bureau of Mines, Minerals Yearbook.
Projections by Charles River Associates.

Policies aimed at encouraging domestic processing of domestic cobalt have the best chance of success, but they may prove expensive. Currently known potential sources of cobalt include domestic primary cobalt ores, such as the old Blackbird Mine in Idaho or the old Madison Mine in Missouri, low-grade nickel laterites, such as at Gasquet Mountain, California, and large nickel-copper sulfides, such as the Duluth gabbro in Minnesota.

Given currently available technology, producing cobalt from domestic primary cobalt ores will probably cost about $25 per pound, half for operating costs and half for annualized capital costs. Even if the United States were to pay all the capital costs, such an operation might not be able to realize a cobalt price high enough to cover its operating costs.

Small quantities of cobalt are currently lost in processing Southeast Missouri lead ores; the U.S. Bureau of Mines and others have investigated the possibility of recovering some of this cobalt. Costs are more uncertain than for the primary cobalt ores. If the operating costs are lower than for the primary cobalt ores, which may not be true, then a partial or complete capital subsidy might be sufficient to encourage continued production in the long run, after the market firms up.

The other options, subsidizing byproduct production from otherwise uneconomic nickel laterites or sulfides, are harder to assess. They seem difficult to justify solely on the grounds of developing a domestic processing industry, although they might be desirable on the grounds of promoting a domestic source of cobalt.

Research and development could play an important role here. Continued efforts by the U.S. Bureau of Mines to investigate methods of processing domestic low-grade laterites, Missouri lead ores, the Duluth gabbro, and primary cobalt ores could pay large dividends in the long run. No one U.S. company may have sufficient financial incentives to do research on the laterites and the Missouri lead ores, because the benefits for any one deposit would be small. The laterites also generally contain small quantities of chromium, an added attraction.

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A

Appendix A

RECOVERY AND PROCESSING OF DEEPSEA NODULES

BACKGROUND

As an ore, manganese nodules pose several problems. Unlike land-based sulfide orebodies, the source of most of the world's nickel, copper, and cobalt today, the nodules are a mixture of oxides. The metals do not occur as separate minerals but are finely disseminated throughout the nodules. In this regard they resemble nickel laterites, but they are much more complex. The nodules average about 25.0 percent manganese, 1.3 percent nickel, 1.25 percent copper, 0.25 percent cobalt, and 0.05 percent molybdenum, by weight (Table A-1). Standard processing techniques for land-based oxide minerals seldom have to recover more than two metals of interest, not four or five. The nodules are highly porous, with water contents of about 30 percent. This is sea.water, so dissolved salts are contained as well. Both the salts and the water must be eliminated in processing. These problems compound the obvious difficulties in developing methods for mining small, fragile objects lying on the sea floor 15,000 feet below the surface.

This appendix will discuss technical and cost factors important in finding, mining, and processing manganese nodules. Attention will be directed to: prospecting and exploration; mining; transportation; processing; and waste disposal. All discussions will assume that an ocean mining operation will process 3 million dry metric tons of nodules annually for 25 years. Four major processes will be described: ammonia leach; hydrochloric acid leach; sulfuric acid leach; and pyrometallurgical. In all of these processes except the hydrochloric acid, manganese recovery can be considered separately from that of other metals. The processes are therefore identified as three-metal or four-metal.

Table A-1

MINERAL CONTENT OF NODULES FROM THE PACIFIC AND ATLANTIC OCEANS
(Percent)

	Pacific Ocean Samples				Atlantic Ocean Samples	
	(1)	(2)	(3)	(4)	(3)	(4)
Manganese	24.2	25.6	29.8	20.6	15.7	7.1
Iron	14.0	5.0	4.8	9.75	15.5	12.5
Copper	0.53	1.2	1.2	0.49	0.14	0.4
Cobalt	0.35	0.18	0.2	0.12	0.41	0.35
Nickel	0.99	1.5	1.36	0.82	0.59	0.08
Silicon	9.4			5.81		1.5
Aluminum	2.9			0.177		2.0
Sodium	2.6		2.6	2.49	2.3	1.0
Calcium	1.9		1.47	1.20	7.32	
Magnesium	1.7		1.7	1.50	1.7	2.8
Potassium	0.8		0.79	0.52	0.31	
Titanium	0.67		0.44	0.43	0.34	0.4
Barium	0.18		0.61	0.05	0.52	0.2
Lead	0.09		0.05		0.15	0.1
Strontium	0.081		0.07		0.19	
Zirconium	0.063					0.04
Vanadium	0.054		0.05		0.07	0.12
Molybdenum	0.052		0.05	0.04	0.05	0.04
Zinc	0.047		0.12		0.05	
Boron	0.029					
Yttrium	0.016					
Lanthanum	0.016					
Ytterbium	0.0031					
Chromium	0.001					
Gallium	0.001					
Scandium	0.001					
Silver	0.0003					
Phosphorus			0.05		0.15	
Lithium				0.01		

Table continued on following page.

Table A-1 (Continued)

MINERAL CONTENT OF NODULES FROM THE PACIFIC AND ATLANTIC OCEANS

(1) Average value for 54 samples, Source: P. H. Cardwell. 1973. "Extractive Metallurgy of Ocean Nodules," Mining Congress Journal. November: 38.

(2) Site in the Pacific Ocean, 10°S, 147°W. Source: P. T. Brooks and D. A. Martin. 1971. "Processing of Manganiferous Sea Nodules." Department of Interior, USBM #7473, January.

(3) J. Kruger and K. H. Schwarz. 1975. "Processing of Manganese Nodules." Metallgesellschaft Review of Activities, Edition 18 -- 1975: Manganese Nodules -- Metals from the Sea.

(4) The Pacific nodule was recovered from water at a depth of 11,254 feet during a Scripps/Mero/U.S. Bureau of Mines-sponsored research cruise using a drag-dredge-type sampler at 20°50'N, 115°12'W. This particular sample had a maximum diameter of 7 cm, which is indicative of it being almost fully sized (since the diameters of the nodules recovered at this location ranged from 1 to 8 cm).

The Atlantic nodule was recovered from water at a depth of 4,000 feet from a sampling station off the South Carolina coast at a Blake Plateau location positioned approximately at 30°51'N, 78°27'W.

SOURCES: Charles River Associates. 1982. Analysis of Major Policy Issues Raised by the Commercial Development of Ocean Manganese Nodules. July. Black, John R. H. 1980. "The Recovery of Metals from Deepsea Manganese Nodules and the Effects on the World Cobalt and Manganese Markets." Ph.D. dissertation, Massachusetts Institute of Technology.

PROSPECTING AND EXPLORATION

In the prospecting stage the ocean mining company uses depth recorders, television cameras, side-scan sonar, and sampling at wide intervals (typically using a grab sampler) to estimate the distribution and grade of nodules over a large area. The aim of the program is to locate areas, not necessarily contiguous, with particularly promising deposits. The prospecting phase is followed by the exploration phase, in which the potential mine sites are surveyed, sampled, and mapped in detail.

MINING

Several methods for collecting nodules and bringing them to the surface have been considered. The most likely approach is to attach a large dredge head to a surface vessel by means of a three-mile long pipestring. Control of such an arrangement poses obvious difficulties, although the design is fairly straightforward. Nodules would be raised to the surface using either standard hydraulic pumps, or an air-lift system. With the air lift, compressed air would be injected into the pipestring at various depths. This reduces the water density within the pipestring, and the pressure differential between the top and the bottom causes water to flow into the bottom of the pipe, carrying the nodules along. The air lift approach would require considerable further development.

One alternative to this system would still use a pipestring, but a mobile mining unit, rather than a dredge head, would collect the nodules. Control of the mining unit would be easier with this approach, but a reliable, self-propelled nodule collecter has yet to be developed.

Other approaches for collecting the nodules and bringing them to the surface seem less likely to prove practical. These include the use of submersibles (i.e., miniature submarines) or the continuous line bucket (an eight-mile loop of nylon rope with buckets attached). This analysis assumes that a dredge head with a hydraulic lift system is used.

TRANSPORTATION

Transportation of the nodules, once they have been brought to the surface, poses some special problems. When they arrive on board the mining vessel, the nodules will be heavily slurried with water. In addition, the nodules are rather fragile, so much of the nodule material is likely to reach the surface as fine particles suspended in the slurry. This slurry must be piped to a storage or transport vessel, then allowed to settle. Complete settling and dewatering will not be possible, so the nodules must be transported to a

processing plant in slurry form. Transport vessels must be designed with these features in mind, to permit them to maintain stability with a semi-liquid, shifting cargo in high seas.

In general, the larger the transport vessel, the lower the transport cost. However, the location of the processing plant becomes an important consideration in selecting a vessel size, because the Panama Canal cannot handle vessels larger than about 55,000 deadweight tons (a medium size vessel, much smaller than the average oil tanker). After reaching the processing plant, the nodules would be reslurried for transporting ashore.

PROCESSING

All of the processes to be discussed for the nodules are similar in some regards, such as in handling the nodules and disposing of solid wastes. The hydrometallurgical methods also resemble each other strongly. In these processes the nodules are leached, separated using liquid ion exchange, and electrowon.

The site-specific capital costs for nodule processing are highly uncertain at present. A suitable site would be located near a deepwater port, have low-cost power available, and permit the nearby disposal of enormous quantities of solids and liquid wastes. Each process has its own special requirements.

PYROMETALLURGICAL

In the pyrometallurgical process, extensively tested by Inco, the nodules would be dried, reduced, smelted, and sulfided, much the way that Inco currently treats its nickel laterite ores in Indonesia. The nickel, copper, and cobalt report to the matte, and the manganese to the smelter slag. Recovery of nickel, copper, and cobalt could then be carried out in standard fashion. If manganese is to be recovered, the manganese-rich smelter slag can be upgraded and processed to produce standard ferromanganese and silicomanganese.

One strength of the pyrometallurgical approach is that it involves few processing risks. The technology for each step is well known. The approach does have problems, however. The nodules must be dried prior to processing, which requires a considerable amount of energy given their high moisture content. It may also have more pollution control problems than the other processes, particularly for gaseous emissions. Finally, manganese recovery from the slag, if desired, may be fairly difficult.

AMMONIA LEACH

This process, developed by Kennecott Copper, involves leaching the nodules with an ammonia solution containing cuprous ions, and simultaneously sparging with carbon monoxide. The cuprous ions act as catalysts to reduce the ore under ambient temperatures and pressures. The nickel, copper, and cobalt are leached very effectively, and the manganese is left behind. Separating the nickel, copper, and cobalt is likely to be carried out using liquid ion exchange, after which the metals would be electrowon to high-purity cathodes. If desired, the manganese-rich tailings could be upgraded to a synthetic manganese ore, then processed into ferromanganese and silicomanganese.

The ammonia leach process has several good features. First, the ammonia used in the process is readily recycled. Second, its energy requirements, at least for the three-metal version of the process, are relatively small. And third, because it is a closed system, pollution control problems are likely to be minor.

Of course, the process is not without some disadvantages. In the four-metal version, the process offers no energy cost savings, because the manganese-rich tailings must be dried before processing. The process is new, so problems are bound to arise in scaling it up to commercial scale. And third, the metal recoveries achieved in pilot plant work were unacceptably low. They could undoubtedly be improved with additional research, however.

HYDROCHLORIC ACID LEACH

The original hydrochloric acid leach process developed by Deepsea Ventures is no longer under serious consideration by companies involved in ocean mining. In this process, dried nodules would be treated with gaseous hydrogen chloride at high temperature, to form manganese chloride and soluble chloride salts of the other metals of interest. The nodules are then sprayed with water, precipitating out iron, and leached with hydrochloric acid. Almost all of the nickel, copper, cobalt, and manganese are dissolved in this process, and almost none of the iron. Liquid ion exchange and electrowinning would be used to recover all four metals from the nodules.

The process is elegant but has a number of potential problems. One is that the world market for electrolytic manganese would be swamped by one such ocean mining operation. The manganese would have to compete with standard high-carbon ferromanganese, or find new markets. Another problem is that manganese recovery is an integral part of the process; it cannot be carried out separately, as in the other processes. Reagent recycling is possible, but not necessarily easy. Depending on where the plant is located, it might be able to arrange to purchase hydrochloric acid from a producer, then sell back the chlorine gas generated in the system. Otherwise, a plant using this process would have to have a ready source of hydrogen (i.e., natural gas) available to recycle the chlorine into acid itself.

SULFURIC ACID LEACH

In this process wet, ground nodules are leached with sulfuric acid under high temperatures and pressures. Nickel, copper, and cobalt are leached into solution, but the manganese remains behind. As with the other processes, the three metals of major interest can be separated using liquid ion exchange, then electrowon. As with the ammoniacal leach process, the manganese-rich tailings can be dried, upgraded to a synthetic ore, and processed into ferromanganese and silicomanganese.

Recycling the sulfuric acid is not practical, so reagent costs for this process are relatively high. The capital costs for the corrosion-resistant autoclaves in the leaching stage are relatively high. The nodules need not be dried in the three-metal version of the process.

PROCESSING TAILINGS TO FERROMANGANESE AND SILICOMANGANESE

The following is a description of the treatment of process tailings that would come from the leaching step in the Ammonia Leach or high-temperature Sulfuric Acid Leach processes. These tailings are rich in manganese and iron and are suitable for upgrading to an impure manganese ore by froth flotation and calcination. By selective reduction, this product could be further processed to a purified manganese ore with a reduced copper, cobalt, and nickel content.

PRODUCTION OF IMPURE MANGANESE ORE

Manganese tailings are composed of individual clear particles of $MnCO_3$ in the case of the Cuprion Process and $MnSO_4$ in the Sulfuric Acid Leach Process, and aggregates of these particles along with particles of clays and iron bearing minerals. Since these particles are either free or in agglomerates, their flotation from the rest of the tailings appears feasible.

The manganese tailings contain about 38 percent solids. Cuprion Process tailings contain approximately 57.1 percent $MnCO_3$, 0.88 percent MnO, 0.22 percent CoO_3, 0.16 percent NiO, 0.13 percent CuO, 33.06 percent inerts, and 8.44 percent combined water. Controlled quantities of the flotation collector, sodium oleate, and soap solution are added with additional water to dilute the tailings to 20 percent solids. The diluted tailings are conditioned for a few minutes, then sent to rougher froth flotation cells. The rougher tailings are sent to a thickener and finally to the tailings pond. The froth is carried to launders and to cleaner flotation cells.

The tailings from this circuit are recycled to the rougher cells, and the flotation concentrate is pumped to another thickener where it is subsequently

filtered in disc filters to contain roughly 55 percent solids. This filtrate is charged to rotary driers which have a thermal load of about 175 million Btu/hr fired by heavy oil (1.5 percent sulfur). From there, the tailings go to oil-fired kilns, each having a thermal load of 180 million Btu/hr. Here the tailings are dried, calcined, and agglomerated. This kiln product is cooled by water sprays and discharged onto a water-cooled conveyor. Dust is removed from the kiln offgases by cyclones and dry electrostatic precipitators before the offgases are discharged to the stack. Drier offgases are treated similarly and then are lime-scrubbed before being sent to the stack. The impure manganese ore contains 59.5 percent manganese, .09 percent copper, .11 percent nickel, and .16 percent cobalt. The recovery efficiency of this process for manganese is expected to be approximately 70 to 80 percent.

For purified ore, the calcined tailings are sent to electric furnaces for selective reduction of the copper, nickel, and cobalt in the ore. This purified ore is 54.3 percent manganese, .03 percent copper, .02 percent nickel, and .02 percent cobalt.

The manganese ore produced by this scheme is of a higher grade than the average 48 percent manganese ores imported into the United States. In addition, an impure iron alloy is produced from electric furnace reduction. It seems most expedient to continue to process this ore to final ferromanganese and silicomanganese products because the marketability of the ore is due to the impurities of copper, nickel, and cobalt. More experimental work needs to be done to ascertain the extent to which impurities can be removed and recovered from the ore by this process.

The consensus among those knowledgeable about the whole nodules processing scheme is that the most economically feasible route is to process the tailings to ferromanganese and silicomanganese final products, because the ferromanganese and silicomanganese, while requiring a large capital investment, bring a higher revenue and a more certain market.

MANGANESE ORE FROM INCO PROCESS SLAG

The manganese-rich slag can be upgraded to a manganese ore by selective reduction in electric furnaces. The contents of the slag have been estimated as follows (Dames and Moore, EIC, 1977):

MnO 43.60 percent (33.8 percent manganese)
Copper 0.23 percent
Nickel 0.09 percent
Cobalt 0.03 percent
Molybdenum 0.07 percent
FeO 4.00 percent
SiO_2 26.00 percent
Al_2O_3 5.00 percent

Manganese ores used for ferromanganese production generally average greater than 35 percent manganese. The more desirable grade is 48 percent manganese. The copper and nickel contents of the slag are higher than in terrestrial ores, so the combined copper and nickel content should be decreased to less than .05 percent. This is achieved by electric furnace reduction. No flotation concentration is involved here.

The slag is charged to electric furnaces along with coke to reduce the copper and nickel. The selective reduction step in the furnaces collects the copper and nickel into an iron alloy. The resulting slag, equivalent to a purified synthetic manganese ore, is then cooled, crushed, and stocked. The estimated analysis of the furnace product is as follows:

Manganese 36.06 percent
Copper 0.03 percent
Nickel 0.02 percent
Cobalt 0.01 percent
Molybdenum 0.01 percent
Iron 0.30 percent
SiO_2 28.72 percent
Al_2O_3 5.62 percent

At 36 percent manganese and 34 percent SiO_2 and Al_2O_3, this is equivalent to a low-grade manganese ore. The desirable requirements for good ore are 48 percent manganese and less than 20 percent of SiO_2 and Al_2O_3 combined.

The purified ore from the leach tailings is similar to a high-grade manganese ore, so it can be treated by conventional technology to obtain ferromanganese. The slag from this step can be further treated to produce silicomanganese, again by conventional technology.

The process consists of charging purified manganese ore to electric furnaces. Coke, lime, and scrap iron are added with the ore to provide the required reducing and fluxing conditions. Ferromanganese containing 78 percent manganese, 5 percent carbon, and the balance iron will be produced. The slag from ferromanganese production is charged to electric furnaces. Coke, lime, and scrap iron again are added to provide reducing and fluxing conditions. Silicomanganese containing 67 percent manganese, 1.8 percent carbon, 17.5 percent silicon, balance iron will be produced.

The ferromanganese from processing the nodules will have to measure up to the ASTM specifications for this product which are: 74-82 percent manganese, 7.5 (maximum) carbon, 1.2 (maximum) silicon, 0.35 (maximum) phosphorus and 0.05 (maximum) sulfur. There are no ASTM limits on nickel, copper, and cobalt but these should be kept low. The most difficult specification to meet will be that for phosphorus. To achieve it, the Mn/P ratio in the slag from smelting should be at least 260, whereas in the nodules themselves it is only about 100.

Sridhar (1976 and 1974) found that the amount of phosphorus left in the pyrometallurgical slag depends on the extent of reduction during smelting. Controlling the reduction to obtain matte containing at least 1.5 percent manganese gives slags with the desired Mn/P ratio. One such slag, described in the previous section, after upgrading contained 0.12 percent phosphorus and was subsequently reduction smelted with added lime at 1620°C. This produced a ferromanganese with 82.0 percent manganese, 5.0 percent carbon, 6.8 percent silicon, 5.8 percent iron, .35 percent combined nickel, copper, and cobalt, 0.04 percent sulfur, and 0.25 percent phosphorus. This meets ASTM specifications except for silicon but this can be remedied by controlling the lime addition, temperature, and amount of reductant, or by subsequent oxidation of the silicon. Sridhar (1976 and 1974) found a 95 percent recovery of manganese in this ferromanganese product. The slag from INCO's smelting process is of lower grade than typical manganese ores. It may need to be combined with higher grade ores and may consume more power per ton of ferromanganese produced. The molten slag, however, can be treated directly, saving a considerable amount of energy over processing leach tailings. The leach tailings contain large amounts of water, which must be eliminated prior to further processing. This reduces some of the energy saved in not drying the nodules at the start of the processing.

Other problems may exist with the alkali content, which is higher for nodules than for normal terrestrial manganese ores. For example, it could create accretion buildup problems in the electric furnaces. Another difference from normal ferromanganese production is the scrap iron charge, not normally needed. It is estimated that about 0.15 tons of scrap per ton of ferromanganese will be needed for processing nodule tailings. Since the production of ferromanganese and silicomanganese is very energy-intensive, the operating costs are very sensitive to the costs of power and coke.

Appendix A

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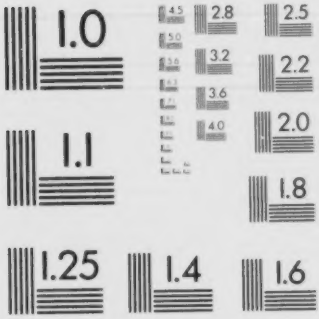
MILLIMETERS

MICROCOPY RESOLUTION TEST CHART
NBS - 1010a
(ANSI and ISO TEST CHART No. 2)



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